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TRANSACTIONS
OF THE
AMERICAN INSTITUTE OF MINING
AND METALLURGICAL ENGINEERS
(INCORPORATED) *and Petroleum*

Volume 196

MINING BRANCH

1953

Metal Mining, Minerals Beneficiation, Coal,
Industrial Minerals, Geology,
Mineral Economics
Geophysics

THIS VOLUME IS MADE UP OF PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS HELD
AT NORMAN, OKLA., OCT. 17-20, 1950; NEW YORK, FEB. 18-21, 1952; HUNTINGTON,
W. VA., JUNE 20, 1952; CHICAGO, SEPT. 3-6, 1952; WHITE SULPHUR SPRINGS,
W. VA., NOV. 14-15, 1952; LOS ANGELES, FEB. 16-19, 1953;
NEW YORK, FEB. 15-18, 1954.

PUBLISHED BY THE INSTITUTE
AT THE OFFICE OF THE SECRETARY
29 WEST 39TH STREET
NEW YORK 18, N. Y.

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FOREWORD

The American Institute of Mining and Metallurgical Engineers presents in this publication of the Transactions of the Mining Branch, Volume 196, seventy-nine technical papers and discussions covering the fields of interest sponsored by the Coal Division, the Minerals Beneficiation Division, the Industrial Minerals Division, the Mining, Geology, and Geophysics Division, and the Mineral Economics Division. In addition, eighteen Extractive Metallurgy papers are listed in the Table of Contents for the convenience of those readers interested in this related field. They appear in the Metals Branch Transactions, Volume 197.

It is believed that the data and results presented are worthy of publication for permanent reference in the Transactions. One important fact persists throughout these various papers. It can be readily seen that there is not only ample opportunity but a definite need for the cooperation and coordination of the various Divisions of the Mining Branch in searching for improvements in the mining and metallurgical treatment of economic ores.

The Mining Branch Council wishes to express its appreciation for the efforts of all who have made this volume possible, not only to those responsible for bringing the papers before the public but also to those authors and researchers who have diligently labored to accumulate these data.

Of particular interest are the four Technical Notes appearing in this volume. These notes offer a special medium of publication for pertinent factual data and specific conclusions arising out of extensive or short investigations that may be a challenge to others to continue or expand specific research. Members of the Mining Branch are urged to avail themselves of this medium of publication.

It is hoped that all readers will benefit from the data here presented and that they in turn will wish to add to the general technical knowledge of the Mining Branch in subsequent Transaction volumes.

LEROY SCHARON, *Chairman*
Mining Branch Council

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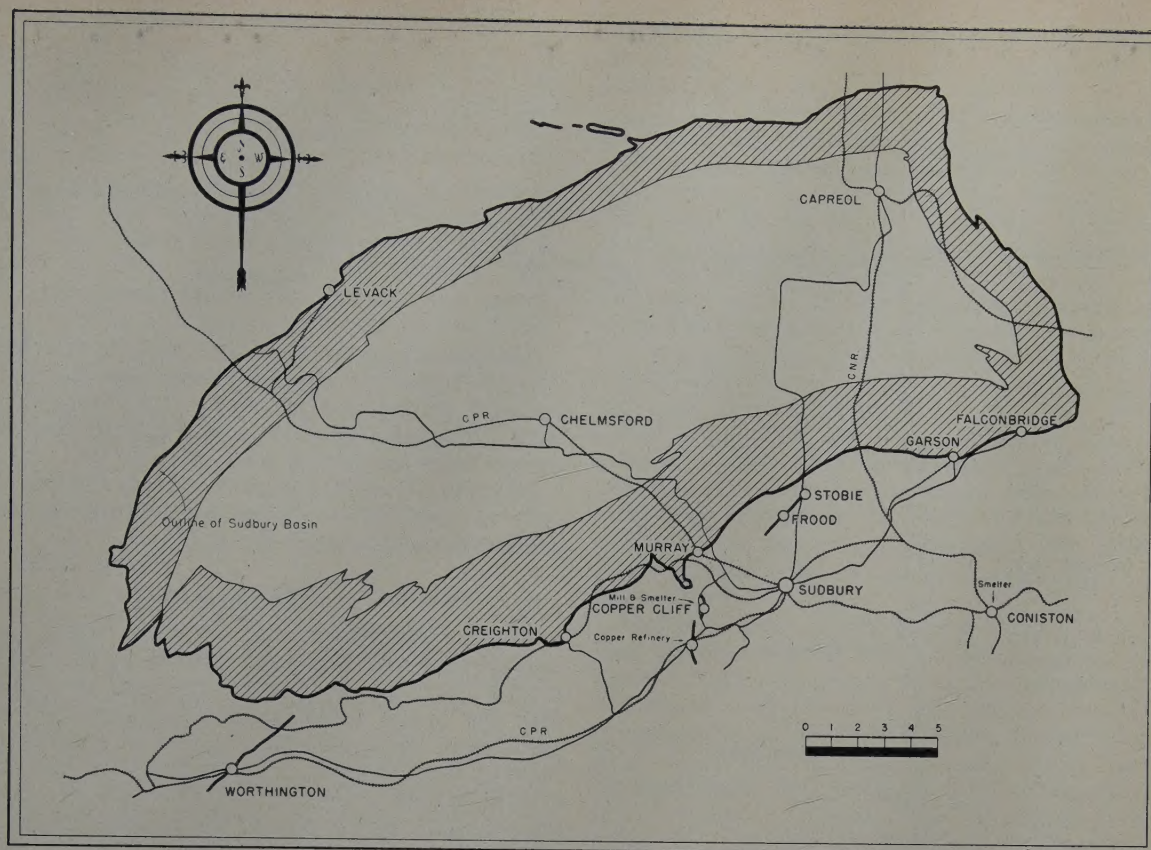
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Extractive Metallurgy

As many members of the Mining Branch are interested in the papers of the Extractive Metallurgy Division, the titles of the papers published for that Division in 1952 are listed below. These papers, and discussion, appear in Transactions (1953) 197, Metals Branch.

- Forging of Arc-Melted Chromium. By H. L. GILBERT, H. A. JOHANSEN, and R. G. NELSON.
- Arc-Furnace Equipment and Its Operation at the Kennecott Utah Refinery. By H. A. SHAW and H. G. G. WHITTON.
- Tristage Crystallization Process for Utilizing Western Ferrophosphorus. By L. H. BANNING, W. E. ANABLE, and R. T. C. RASMUSSEN.
- Kinetics of the Oxidation of Galena in Sodium Hydroxide Solutions under Oxygen Pressure. By J. E. ANDERSEN, J. HALPERN, and C. S. SAMIS. With discussion.
- Operations at New Cornelia Copper Smelter of Phelps Dodge Corporation. By J. W. BYRKIT.
- Effect of Dissolved Sulphur on the Surface Tension of Liquid Copper. By C. F. BAES, JR. and H. H. KELLOGG.
- Free Energy of Vaporization of Metals from 0° to 2000°C. By J. W. EVANS.
- Vapor Pressure of Zinc in the Reduction of ZnS by Cu and Fe. By A. W. BETHUNE and L. M. PIDGEON. With discussion.
- Vacuum Dezincing of Desilverized Lead Bullion. By T. R. A. DAVEY.
- Thermodynamic Properties of Molybdenum Dioxide. By N. A. GOKCEN.
- Constitution of the FeO-Fe₂O₃-SiO₂ System at Slagmaking Temperatures. By R. SCHUHMAN, JR., R. G. POWELL, and E. J. MICHAL.
- Solutions of Metals in Fused Salts. By DANIEL CUBICCIOTTI.
- Reaction of Silver with Aqueous Solutions of Cyanide and Oxygen. By G. A. DEITZ and J. HALPERN.
- System Ag₂O-B₂O₃: Its Thermodynamic Properties as a Slag Model. By G. M. WILLIS and F. L. HENNESSY.
- Effect of Chloride on the Deposition of Copper, in the Presence of Arsenic, Antimony, and Bismuth. By V. HOSPADARUK and C. A. WINKLER.
- Hurley Furnace and Boiler Description and Design. By E. A. SLOVER.
- Preparation of Metallic Iron of High Purity. By G. A. MOORE. With discussion.
- Sintering Zinc Concentrates on the Blackwell 12 by 168 Ft Machine. By A. E. LEE, JR.



Underground Mining Methods At International Nickel Company

by H. J. Mutz, A. F. Brock, and W. J. Taylor

THE International Nickel Co. of Canada Ltd. operates five underground mines and an open pit. Four of the mines, the Frood-Stobie, Creighton, Murray, and Garson, are on the south range of the Sudbury Basin, map above, within a radius of 10 miles of the city of Sudbury, Ontario. The fifth mine, the Levack, is on the north range of the Sudbury Basin, 30 miles to the northwest of the city. The open pit is operating on surface ores of the Frood-Stobie orebody.

Two pits were opened at the Frood-Stobie mine; the Frood section was started in 1938 and the Stobie section in 1942. To meet the heavy demand for nickel during World War II, open-pit production was accelerated until it constituted over 40 pct of Inco's total tonnage. As a consequence open-pit ore was depleted considerably ahead of schedule; the Stobie pit was completed in the summer of 1951, and it is

H. J. MUTZ, Member AIME, is Superintendent of Mines, and A. F. BROCK and W. J. TAYLOR are Operating Engineer and Planning Engineer, respectively, International Nickel Co., Copper Cliff, Ont., Canada. In preparing this paper they were assisted by members of the mining staff at International Nickel.

Discussion on this paper, TP 3461A, may be sent (2 copies) to AIME before March 31, 1953. Manuscript, April 14, 1952. New York Meeting, February 1952.

expected that mining at the Frood pit will be completed in 1953.

The major program launched 10 years ago to develop additional underground production as a replacement of open-pit tonnage thus had to be greatly speeded up. This conversion to all-underground mining will be completed within the next two years, and the underground mines will then have an annual hoisting capacity of 13,000,000 tons.

One of the new mines in the underground expansion program, the Murray, was brought into production in 1950. Ore production from the new Stobie section of the Frood-Stobie mine was commenced on a limited scale in 1951 and will be increased as the program progresses. Delivery of ore to the new Creighton mill was started in the summer of 1951 from the caving project launched at Creighton mine.

When mining was started in the Sudbury district 65 years ago the ore deposits were opened on a small scale and mining methods conformed to a general pattern of small open cast pits, followed by open stopes as mining was extended below the pits. The open stoping methods used were heading-and-bench and shrinkage. In certain cases a type of cut-and-fill mining was used with drywall tramways and dry-wall chutes in the fill. As the mines were deepened,

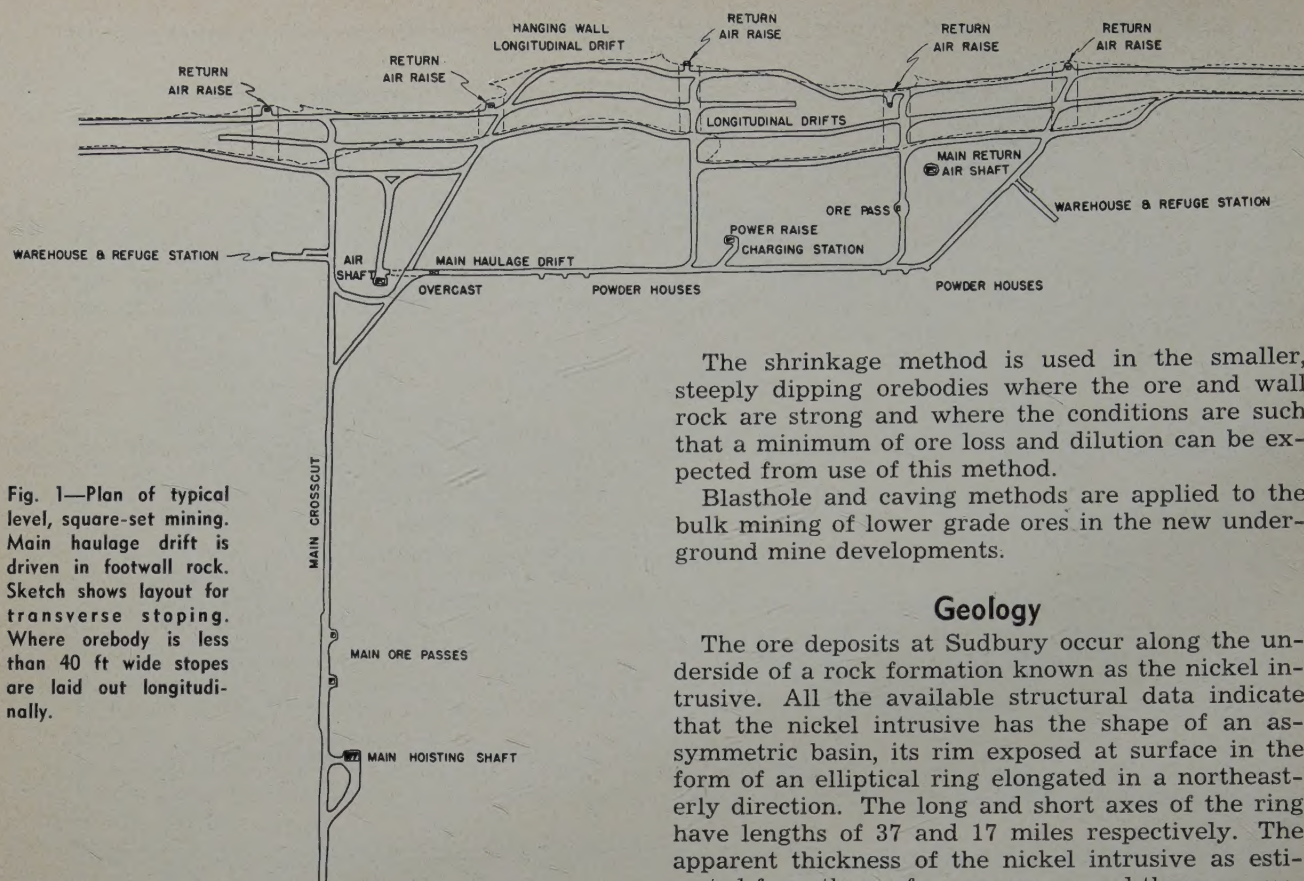


Fig. 1—Plan of typical level, square-set mining. Main haulage drift is driven in footwall rock. Sketch shows layout for transverse stoping. Where orebody is less than 40 ft wide stopes are laid out longitudinally.

a general change to cut-and-fill and square-set mining was effected, and these methods became standard for mining the higher grade ores of the district. With the improvement of metallurgical practice and the development of low-cost bulk mining methods, recovery of the lower grade ores became economically feasible. Blasthole and caving methods are used to mine these low grade ores.

The methods of mining now employed at Inco mines are square-set, cut-and-fill, shrinkage, blasthole, and induced caving. Blasthole and square-set methods are used at Frood-Stobie, square-set and caving at Creighton, blasthole at Murray, shrinkage, cut-and-fill, and square-set at Garson, and blasthole and square-set at the Levack mine.

The choice of stoping methods at each of the mines is determined by conditions peculiar not only to the individual mine, but also to the particular areas and horizons of each mine: the size and shape, attitude, composition, and grade of the orebody; the inherent physical properties of the host rock; the extent to which both the ore and the host rock have been weakened by faulting and movement; the depth below surface; the necessity to minimize subsidence and control rock pressure. In each instance these factors have a bearing on the stoping method which is selected.

Square-set and cut-and-fill methods are used 1—to obtain complete extraction with a minimum of dilution of the higher grade ores and 2—to minimize caving and subsidence of the overlying ground. The square-set method is used where both ore and wall rock are weak, cut-and-fill where the wall rock may be weak but the ore itself strong. The square-set method is also used to recover vertical and horizontal pillars in filled stope areas, as well as to recover remnants of ore left in stopes or in the fringes of orebodies that have been worked by other methods.

The shrinkage method is used in the smaller, steeply dipping orebodies where the ore and wall rock are strong and where the conditions are such that a minimum of ore loss and dilution can be expected from use of this method.

Blasthole and caving methods are applied to the bulk mining of lower grade ores in the new underground mine developments.

Geology

The ore deposits at Sudbury occur along the underside of a rock formation known as the nickel intrusive. All the available structural data indicate that the nickel intrusive has the shape of an asymmetric basin, its rim exposed at surface in the form of an elliptical ring elongated in a northeasterly direction. The long and short axes of the ring have lengths of 37 and 17 miles respectively. The apparent thickness of the nickel intrusive as estimated from the surface exposures and the near surface dips ranges from 1 to 2 miles. It consists of an upper layer of micropegmatite separated by transition rock from a lower, somewhat narrower layer of norite.

The space inside the basin structure is occupied by a series of volcanic and sedimentary rocks, the layering of which outcrops in concentric rings inside those of the nickel intrusive, indicating for this series a basin shape conformable with that of the nickel intrusive. The rocks underlying the intrusive consist, in order of abundance, of gneisses, volcanics, sediments, granite, and breccias.

It appears that an erosional unconformity existed between the rocks that now overlie, and the rocks that now underlie the nickel intrusive, and it is generally accepted, although not proved, that the nickel intrusive was emplaced horizontally between flat-lying rocks above the unconformity, and highly deformed rocks below the unconformity. Differentiation of the nickel intrusive into its component parts, followed by down-warping or infolding, complete the steps by which the basin structure is thought to have been formed. Post-nickel intrusive dikes of granite, trap, and olivine diabase intersect the basin structure, and its continuity is interrupted by faults that displace contacts for distances ranging from a few feet to $2\frac{1}{2}$ miles.

Mining experience of more than 50 years has established the bottom contact of the norite as the primary structure that determined the location of ore deposition. Feeble mineralization along this locus is present for most of its known area, but ore concentrations occur only where the primary locus coincides with essential but secondary structures. The recognized kinds of ore-bearing structure are, first, depressions in the relatively smooth norite footwall in which the norite penetrates the underlying rocks, and second, shearing that roughly coincides with the base of the norite. Either of these

structures is capable of controlling an ore deposit, or both may be present in one deposit; however, it should be noted that although every known orebody is associated with the kind of structures described, there are many such structures in which the mineralization is not of ore grade.

Numerous attempts have been made to explain the obviously close relationship of the ore mineralization to the norite. Their invariable association indicates a common source, and adherence of the ore to the base of the norite has led many geologists to believe that the ores must have accumulated from the nickel intrusive during the process of differentiation. Others have given more weight to the apparently contradictory evidence based on the timing of geological events, and have proposed that the ores were introduced hydrothermally after the nickel intrusive was frozen and deformed. The balance of evidence favors the first explanation, but in the final analysis it may prove that neither explanation is wholly correct.

The ores are mined primarily for nickel and copper, but there is also important production of cobalt and of the platinum group metals, as well as gold, silver, selenium, and tellurium. The principal ore minerals are pyrrhotite, pentlandite, and chalcopyrite; pyrite and the arsenides of nickel, cobalt, and platinum are the notable accessory minerals. The typical orebody lies astride the norite contact and consists of varying proportions of disseminated ore, breccia ore, and stringer ore. Most of the sulphide on the norite side of the contact occurs in the disseminated form, whereas the sulphide below the norite contact is massive, although it always contains some associated rock; it is named *breccia* ore or *stringer* ore, depending on whether the sulphide or the rock is the continuous component.

Levack mine is located on the flatter northwest side, and the other four mines on the steeper southeast side of the assymetric basin. Levack, Murray, and Frood-Stobie are examples of deposits where the secondary structural control consists of depressions in the base of the norite, or they might be described as places where the norite has made unusual penetrations into the footwall rocks. At Levack and Murray the penetrations take the form of blunt embayments or troughs with long dimensions running down an approximate 40° dip at right angles to the norite contact. The Murray orebody is a single mining unit. The Levack mine consists of a group of orebodies separated by nonmineable mineralization along the strike and overlapping each other in depth range.

The Frood-Stobie mine is a special example of ore control by a depression in the norite footwall. In this instance the norite penetration of the underlying rocks, instead of being blunt, is sharp and very deep. It has the shape of a dike-like appendage projecting downward from the bottom of the norite. Erosion has removed the horizons at which continuity once existed between the appendage and the main body of norite, so that the Frood-Stobie orebody occurs in a pocket of norite now isolated in the basement rocks about a mile from the parent body. At the surface elevation this remnant has an overall length of 9600 ft. In vertical cross section it wedges out downward along a 75° dip and bottoms, for the length so far explored, at about 4000 ft. All the norite in the remnant carries disseminated sulphides, and although the intensity varies widely, most of the norite is mineable; in addition, there is

a large, but by comparison, minor volume of breccia and stringer ore located along the bottom of the deepest parts of the disseminated unit.

The essential secondary structure at Garson is shearing that coincides approximately with the norite contact. The dominant shearing strikes east-west and dips at about 75° to the south, away from the center of the basin. A number of subsidiary shears striking east-west and dipping about 55° to the south diverge upward from the parent structure from successively deeper horizons and terminate in the norite. In vertical cross section the norite contact traces an irregular course across each of the successively deeper wedges of ground lying between the main shear and its flatter branches. The ore occurs along the main and tributary shears and along the norite contact between them; hence the overall mining pattern is complex. Mineable widths vary from several feet in the vein-like deposits of ore along the shears up to a maximum of about 200 ft where swells of ore occur in the zones of shear convergence.

The structural control for the Creighton ore consists of both shearing and a depression in the norite footwall. At surface the embayment formed by the depression is about a mile long and half a mile deep; it plunges at 45° to the northwest. A northwest dipping shear strikes into the depression from the northeast and there develops a branching pattern so oriented that it coincides with the plunge of the depression. The various branches of the shear cut across the southeast side of the depression where they diverge downward and southeastward. Consequently each branch, along both the strike and the dip, passes from norite into the footwall rocks and causes displacements of the norite contact ranging up to several hundreds of feet.

The mineralized zone has the shape of an irregular but continuous pipe lying along the plunging structure just described. Most of the zone consists of norite containing disseminated sulphides, the

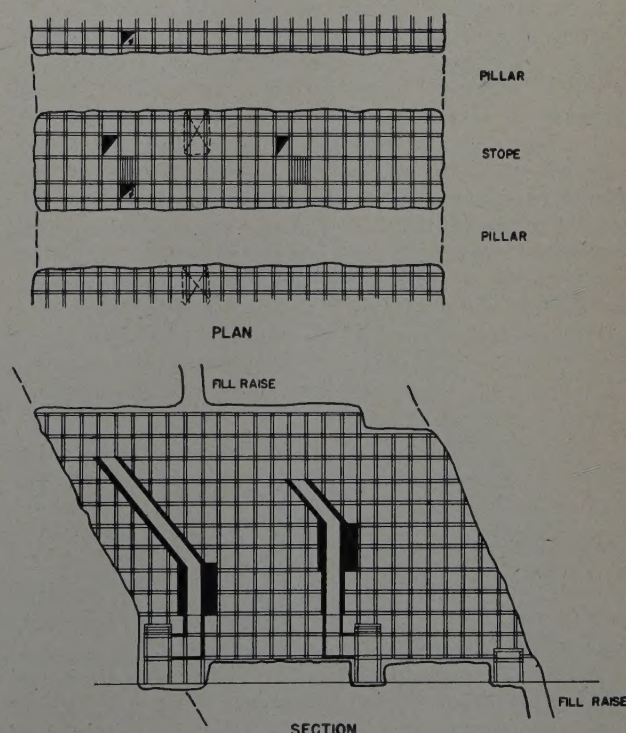


Fig. 2—Plan and section of square-set stope.

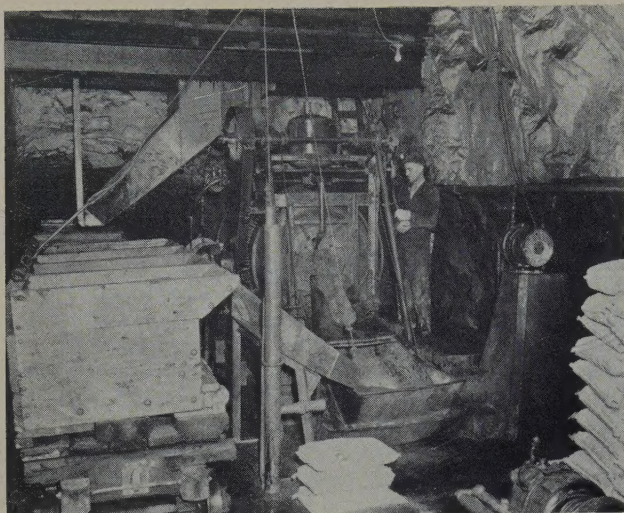


Fig. 3—Concrete mixing station.

minor portion being breccia and stringer ore located in large and small pockets on the underside of the mineralized norite and in sheets along the shears. In a general way the intensity of the disseminated mineralization decreases upward from the base of the norite, but heavier than normal disseminations occur adjacent to the sheets of breccia ore that lie along the shears.

The early mining, dependent on direct smelting ores, was confined to the breccia ore deposits. In more recent years the introduction of milling and the development of suitable mining methods have permitted the mining of an ever-increasing proportion of the disseminated norite.

Stoping Methods

Square Set

General Layout: Narrow square-set stopes and pillars have been found the most satisfactory from the standpoint of safety and ground control, and a system of transverse stopes $30\frac{1}{2}$ ft wide with 19-ft rib pillars is used where an orebody is more than 40 ft from footwall to hanging wall. In areas where an orebody is less than 40 ft wide the stopes are laid out longitudinally; the length of a standard longitudinal stope is equivalent to the width of two standard transverse stopes and one pillar. The pillars in longitudinal stoping areas are the same width as those between the transverse stopes.

The level development, see Fig. 1, for transverse square-set stoping consists of a main haulage drift in the footwall rock, one or more 11x11-ft longitudinal extraction drifts, and an 11x11-ft longitudinal return air and supply drift for the stopes being mined on the level below.

The extraction drifts are in ore and the number driven is governed by the width of the orebody. The first is driven on the footwall contact, and where more than one is required the others are parallel to it at approximately 50-ft intervals. The return air and supply drift is also in ore, close to the hanging wall. Crosscuts at 500-ft intervals connect the footwall drift with the drifts in the orebody.

In normal ground extraction drifts are timbered as they are driven, with temporary caps and lagging for back support, and slashed later to a height of 18 ft to make room for permanent gangway timber.

The permanent gangways are two sets high. The sets are $5\frac{1}{2} \times 8\frac{1}{2}$ ft center to center of posts, and the clear width inside the gangway is 7 ft 8 in. The clearance above the base of rail in the lower sets is 7 ft 5 in., and the inside height of the upper sets is 6 ft 6 in. As the upper gangway sets are erected they are lagged with 8-in. flatted timber. The lagging is supported on false caps over the gangway caps and separated from them by squeeze blocks. The permanent gangway covering is put on after the sill has been cut in the stope above.

Where the extraction drifts cannot be opened to full height without more support than is afforded by the temporary back timber, they are timbered as they are driven with lower floor gangway sets. The back slash to make room for the second floor sets is broken down on 8-in. lagging placed on the sets before blasting.

The return air drifts are timbered with lower floor gangway sets, and 3-in plank is used for back lagging over the sets. Entrance to the stope manways is from the extraction drifts, and the rounds for the manway bases are driven in the drift wall before standing the permanent timber.

All square-set stopes are silled 9 ft above the base of rail of the level and are mined to within 50 ft of the level above, leaving a 59-ft crown pillar



Fig. 4—Lever-operated gangway chute.

that is not recovered until the pillars above it have been mined out.

A 7x11-ft cribbed stope raise to serve as a manway, airway, pipeway, and supplyway into the stope is driven from the back of the sill floor to the hanging wall longitudinal drift on the level above. The raise is collared midway between the footwall and hanging wall and adjoining the pillar on the side of the stope opposite the manway. Vertical raises are preferred, but they may be driven at any inclination down to 55° .

Stope Timbering: Originally all stopes were timbered with sets $5\frac{1}{2}$ ft square, and five sets were used to span from pillar to pillar in a standard transverse stope. This is still the practice in unusually heavy ground. In normal ground present practice is to use four sets to span the stope, a $5\frac{1}{2}$ -ft square set against each pillar with two $8\frac{1}{4} \times 5\frac{1}{2}$ -ft sets between.

The sets used in stopes are framed in the mine shops from local pine timber cut to exact length at the mills and flatted on two sides to 8 in. thick.

The use of concrete in constructing chutes has made it possible to handle all the ore broken in a transverse stope up to 100 ft long, or approximately 50,000 tons, through one chute.

In the four-set stopes the chute and a wing from one side of the chute open into adjoining sets in the two center rows. In five-set stopes the chute is in a set in the center row and has wings from two sides into each of the adjoining sets in the rows on each side of the center.

With only one chute in a stope the chute is spaced about halfway between the hanging wall and footwall to give a slushing distance of not more than 50 ft. The chute is inclined parallel to the dip of the footwall to maintain its position in relation to the footwall on succeeding floors as the stope is mined.

Two or more chutes, see Fig. 2, are required in stopes where the orebody is wider than 100 ft. One of the chutes is kept 15 ft from the footwall and the others are spaced from it at approximately 50-ft centers. Normally the chutes are carried up directly from the gangway in the longitudinal drifts, but where an extreme flattening of the footwall occurs close to the haulage level, boxholes up to 60 ft long may be driven in waste rock to the required position of the footwall chute in the stope.

One manway only is carried up from the level in each stope. The manway is in the row of sets next to a pillar, and when the stope has been mined out the manway is disconnected at the top to the stope raise for subsequent use as a service opening to the adjoining rib pillar.

The stope ore chutes open into the upper gangway sets in the longitudinal haulage drifts and from there to the top of the second floor of the stope the chutes are constructed of timber. From the second floor to within four floors of the top of the stope they are concrete. The timbered portion of the chute is 5 ft square inside, and the walls are built up of heavy salvage timbers not less than 3 ft in length packed end-on to the inside face of the chute to form solid panels. The timbers forming the panel are spiked together as they are laid, and the panels are secured by wedging against the stope square sets. Timber is used for this construction in preference to concrete because it is more resistant than concrete to blasting.

The concrete portion of the chute is built up through the square sets and is poured in 7-ft vertical lifts corresponding to the standard square-set height. To allow the use of chute wings the top of the chute is kept one set below the mucking floor and each new lift is poured before filling is started on the floor above.

In the construction of concrete chutes the outside forms are made of 1-in. rough lumber nailed to the square set timbers. Inside forms are prefabricated and consist of an angle iron frame and rectangular panels made of 1¼-in. dressed fir. The forms are supported on a chute platform held in position with four 1¼-in. diam pins placed in 2-in. pipe sleeves set in the top of the previous pour. Wedges driven into the corners between the panels hold them securely against the angle iron frame.

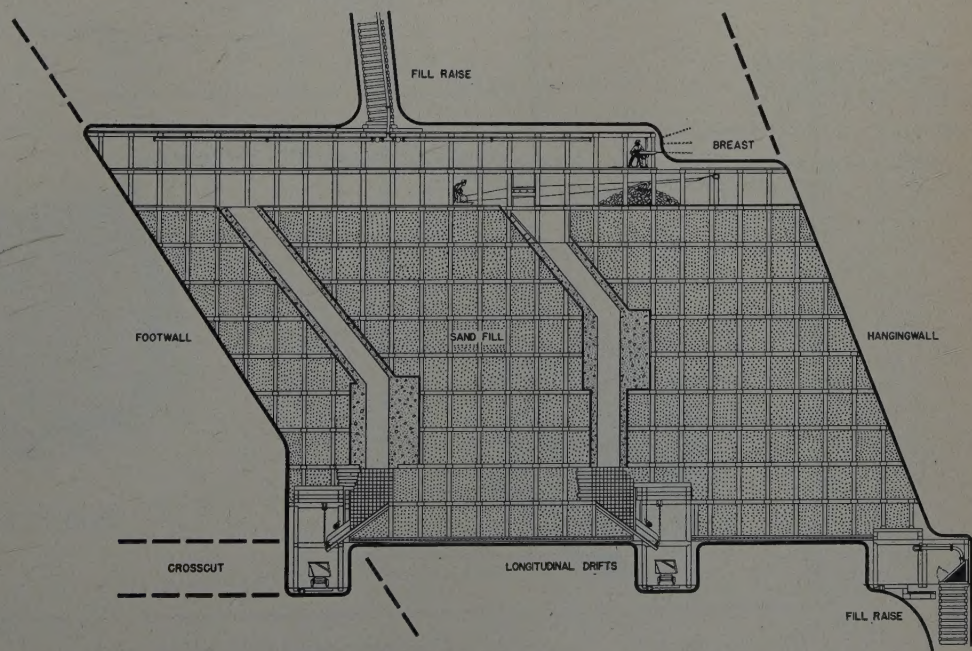
The concrete portion of the chute is 4½ ft square inside. The walls are 14 in. thick in the vertical portion; in the inclined portion the bottom is 18 in. thick, the side walls 15 in. thick, and the top 14 in. thick. Where a change in inclination from the vertical to an incline is made, the hanging wall and footwall sides of the vertical portion are strengthened by increasing the thickness of the concrete for 14 ft below the point of change.

Concrete is mixed at a central mixing station, see Fig. 3, and is delivered to the stope from the level above through a salvage wood-stave pipeline in the stope raise. The aggregate used is pit gravel with boulders more than 3 in. in diam screened out, and the mix contains one part cement to five parts gravel with a dispersing agent added to reduce segregation during transport.

Description of Mining Cycle

Stoping: Stope silling is begun by driving a 10x10-ft pilot drift 9 ft above the base of rail on the longitudinal center line of the stope from one of the longitudinal extraction drifts to the stope limits at the hanging wall and footwall. Subsequent widening from the pilot drift to the pillar walls is done by diamond-drill blastholing. A diamond drill is set up on the second floor of the gangway in an extraction drift, and blastholes are drilled parallel to the pilot drift to break a slice 3½ ft thick extending

Fig. 5—Section through a standard square-set stope, which is mined overhand in a series of 7-ft high horizontal slices. New slices are started at the stope raise. A slot two sets wide is opened across full stope width. From the slot the floor advances first to hanging wall, then to footwall in a series of blasts equivalent to a complete row of sets.



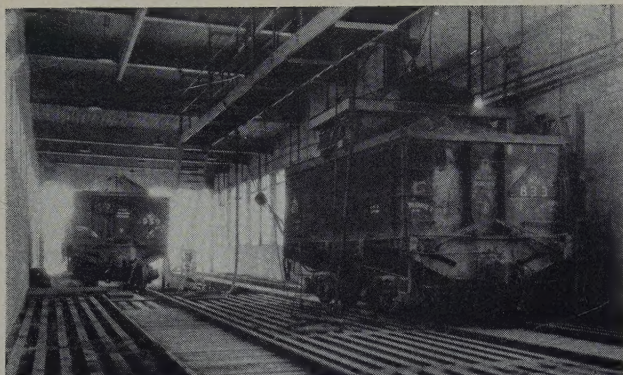


Fig. 6—Unloading tracks at Froid-Stobie sand plant.

from one extraction drift to the next, an approximate length of 50 ft. Four blastholes, at 3-ft centers vertically, are drilled for each slice. They are column-loaded with $1\frac{1}{4}$ x8-in. sticks of 75 pct gelatin dynamite and are detonated with delay electric blasting caps. Three successive slices are blasted, and the broken ore from each slice is removed before the succeeding slice is blasted. Widening in the first block selected is completed, and if necessary this portion of the stope is timbered, before widening on the opposite side. Adjoining blocks are widened in a similar manner until the stope is completely silled. The ore broken on the sill is removed by slushing into cars in the extraction drifts with 15-hp double-drum, air-operated slusher hoists and 42-in. scrapers.

In heavy ground power-feed drifters are used instead of diamond drills, and $10\frac{1}{2}$ -ft holes at $2\frac{1}{2}$ -ft centers are drilled to break a 10x10-ft slash into the pilot drift. The broken ore from each slash is removed immediately and the open ground is timbered before further drilling is done.

The stope fill raise is started when silling has been finished around the raise location. An area 3 sets square is mined on the second floor and the raise is collared in two of these sets. It is driven to the level above while the remainder of the stope is being silled and the second cut is being mined. A slide is built from the raise to the nearest extraction drift to allow the broken ore from the raise to be loaded directly into cars.

Sill floor timbering is started at the extraction drift gangway and the caps and girts of the sill square sets are placed to match those of the second gangway sets. The sill sets stand on the solid floor of the stope; the post length varies with the level of the floor and on this account the posts are supplied overlength, framed on one end only, and are trimmed in the stope to the proper length.

Mining the second floor of the stope follows the completion of silling. After the broken ore from the second cut has been removed the sill is prepared for filling; extraction drift gangways are sheeted and lagged, loading chutes, see Fig. 4, are built and equipped with lever-operated control gates, chutes are raised from the second floor of the gangway to the top of the sill sets, and the permanent sill floor is laid and pillar fences are built. The floor consists of 5-in. flatted 11-ft sills placed lengthwise of the stope at $5\frac{1}{2}$ -ft centers and covered with two layers of 2 in. x 11 ft plank. The plank in the upper layer is placed at right angles to that in the lower layer, and joints between planks in both layers are staggered. For the pillar fence 2 in. x 11 ft plank or 11-ft edged slabs are nailed to the inside face of the square set posts along each pillar wall. Joints between planks of adjoining rows are made on alternate posts. The extraction drift gangways are sheeted with 8x10-in. timbers 9 ft 4 in. long laid on two 10x12-in. longitudinal stringers, one above each line of gangway posts, and supported on and separated from the gangway sets by 6x10-in.

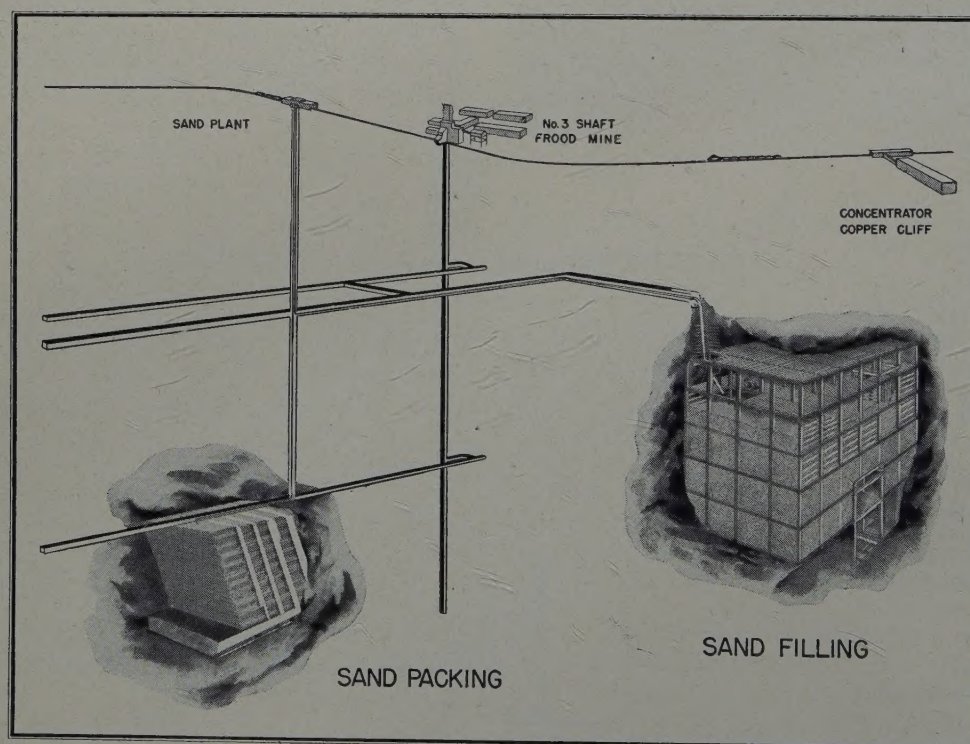


Fig. 7—Diagrammatic sand fill flowsheet.

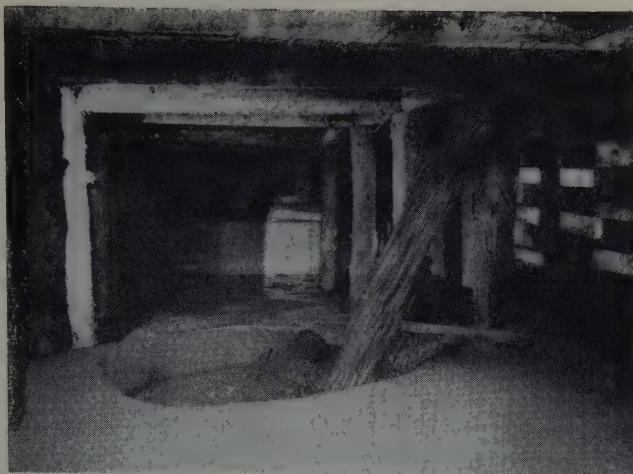


Fig. 8—Pouring sand fill in square-set stope.

squeeze blocks 15-in. long placed over the posts. The sides of the gangway are lagged by nailing plank to false posts separated from the gangway timber by 8-in. spacer blocks.

In a stope to be filled with water-borne sand, all openings from the drifts into the stope are sealed with a single layer of burlap which is extended 2 ft away from the sides of the gangway into the stope before being cemented to the floor and walls.

A stope, shown in Fig. 5, is mined overhead in a succession of horizontal slices each 7 ft high. Normally each new slice is started at the stope raise. Before starting the cut, standard practice is to pin the raise cribbing to the solid 7 ft above the back of the stope, cut an opening through the side of the crib 21 ft above the back, block the chute compartment at this point with a platform supported on booms, and then remove the cribbing from the bottom 7 ft of the raise and widen this portion to make room for two sets of timber. Two more sets are broken to form a slot two sets wide that is then opened across the full width of the stope. From this slot the mining floor is advanced first to the hanging wall and then to the footwall in a series of blasts each equivalent to one complete row of sets.

Drilling in the stopes is done with power-feed drifters on 24-in. sliding cone shells using 1-in. quarter-octagon drill rods with detachable bits. Standard column bars are 5½ ft long and column arms are 5-ft lengths of 3-in. extra heavy pipe with universal clamps and 5-in. cone dumps.

The number of holes required to break a breast 30½ ft wide varies between 30 and 40. Holes are 6 ft deep and drilled in three rows. In the upper row the holes are collared below the girts and are inclined upwards at 15°; the pillar holes in each row are collared inside the posts and are drilled outward at 15°.

The standard explosive for blasting in stopes is 70 pct semi-gelatin dynamite, and the normal charge is three to four sticks per hole. Each stick of explosive is tamped separately, and the holes are stemmed with 1¼x8-in. sticks of clay stemming. The holes are detonated with No. 6 blasting caps and tape fuse supplied in 10-ft lengths and trimmed to give the proper firing order.

Before the blast is fired, lagging under the breast and the front half of the mining floor sets next to the breast is removed. The sets are well braced with angle braces and all exposed members of sets beneath the breast are covered with bulkhead timber.

After each breast has been blasted the back and breast are washed, inspected for missed or cut-off holes, and scaled from under cover of the last row of sets. The area adjacent to the blast is cleaned up, and broken or dislodged timber is replaced. The booms are then advanced, blocked, and covered to give protection to the men working near the breast on the mining floor and on the slushing floor below. Slushing is started as soon as it is safe to do so, and when enough ore has been removed to make room for timber the timbermen stand the new line of sets. The set timbers are fastened together as they are erected and when the row has been completed are blocked to the pillar with 8-in. flatted timber and to the breast with 5-in. timber.

Blocking on the booms is caught up on the sets by placing a 3-in. plank and a 7-in. bulkhead timber over each cap and then wedging between this timber and the cross blocking on the booms until the booms are freed. Booms are not taken down but

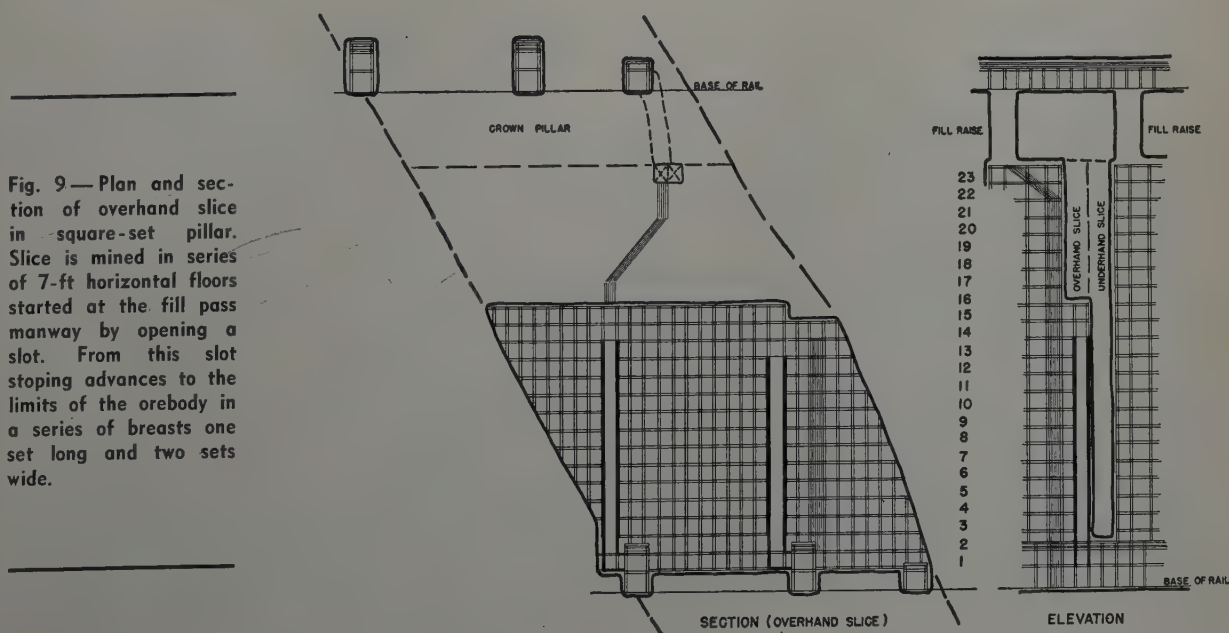


Fig. 9—Plan and section of overhead slice in square-set pillar. Slice is mined in series of 7-ft horizontal floors started at the fill pass manway by opening a slot. From this slot stoping advances to the limits of the orebody in a series of breasts one set long and two sets wide.



Fig. 10—Concrete floor in cut-and-fill stope.



Fig. 12—Air-leg drill in cut-and-fill stope.

are left in place ready to be moved after the next blast. Each set is lagged with 5-ft slabs.

For handling the broken ore in the modified 4-set stopes 15-hp double-drum air-operated slushers and 42-in. scrapers are standard equipment; 7½-hp double-drum slushers are used in 5-set stopes. In the 4-set stopes the ore is slushed in the two center rows of wide sets into the chute or wing. The floor in the slusherways is 60-lb rail at 15½-in. centers; along each pillar wall the sets are floored with 2-in. plank. Slides of 5-in. flatted timber built beneath the breast in the two outer rows of sets deflect the broken ore into the slusherways and also serve as travelways under the breast.

Chute openings are guarded by a grizzly of 60-lb. rails at 15½-in. spacing raised 2 ft above the floor of the stope. Ore is scraped into the chute through the end opening between the floor and the grizzly and pieces too large to go through this opening are blockholed.

Filling

The most recent and significant development in the square-set method of stoping at Inco has been the use of water-borne sand at Frood-Stobie and

Creighton for sand filling current square-set stopes and for sand packing the older waste-filled stopes to consolidate the fill and stabilize the ground left in pillars.

The screen analysis of the fill sand is:

Mesh	Pct
+ 65	12
— 65 and +200	60
—200 and +800	25
—800	3

Sand for Frood-Stobie, recovered from tailing at the concentrator in Copper Cliff, is dewatered and shipped to the mine in railway cars.

At the mine the sand is dumped into concrete receiving bins, see Fig. 6, and repulped for use underground to a density of 60 to 65 pct solids by hydraulic monitors supplied with recirculated mine water at 100 psi. The pulp is flushed from the bins into a rake agitator and is drawn from the agitator tank into a distributor box with connections to three sand pipelines that enter the mine through a

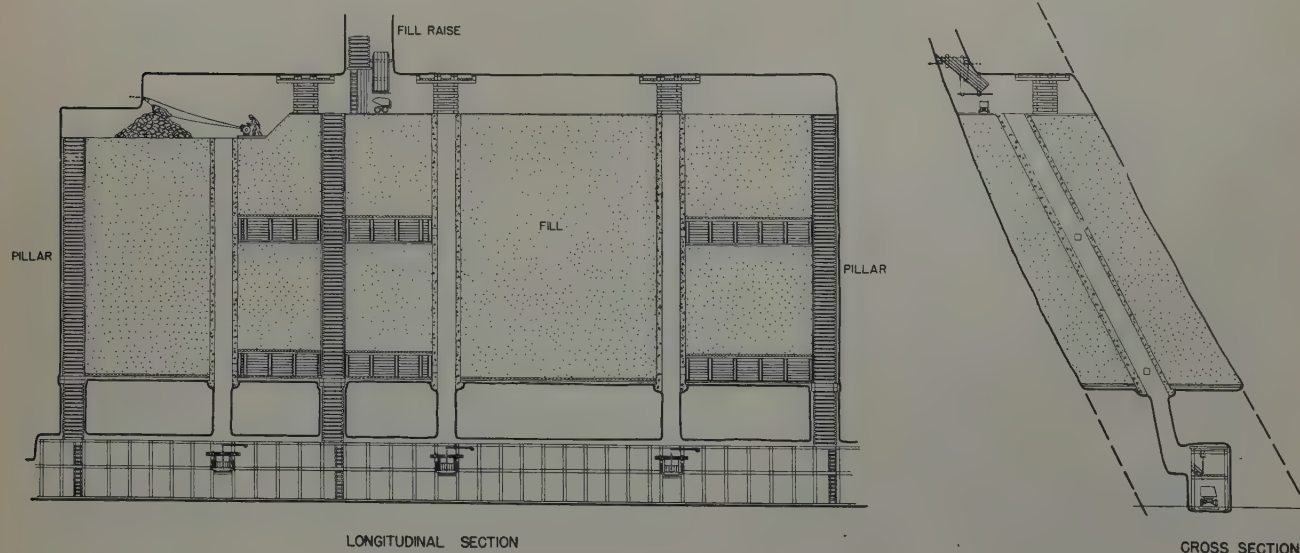


Fig. 11—Sections of a longitudinal cut-and-fill stope.



Fig. 13—A 15-hp slusher in cut-and-fill stope.

service raise extending from 3100 level to surface. A diagram of the method of distribution is shown in Fig. 7. The pipe used for the sand mains through the service raise and the main haulage drifts on the levels down to 1800 level is 6-in. standard weight rubber-lined pipe with 400-lb loose flanges, and from the 1800 to the 3100 ft levels is 6-in. extra heavy rubber-lined pipe with 600-lb loose flanges.

Distribution lines on the levels from the mains to the stope raises are of unlined 4-in. extra heavy pipe, with 400-lb flanges to within 1500 ft of the end of each distribution line and 150-lb flanges for the rest of the line.

No valves are used; each line from surface serves one level at a time and the connection from a main in the service raise to the level line is made by removing one length of straight pipe from the main and connecting in a bend to join the two lines. Connections from the 4-in. pipelines on the levels to the lines in the stope raises are made with lengths of 4-in. wire-wound hose with flanges of hard rubber connected to the pipelines by split steel flanges.

An independent telephone system provides direct communication between the working place under-

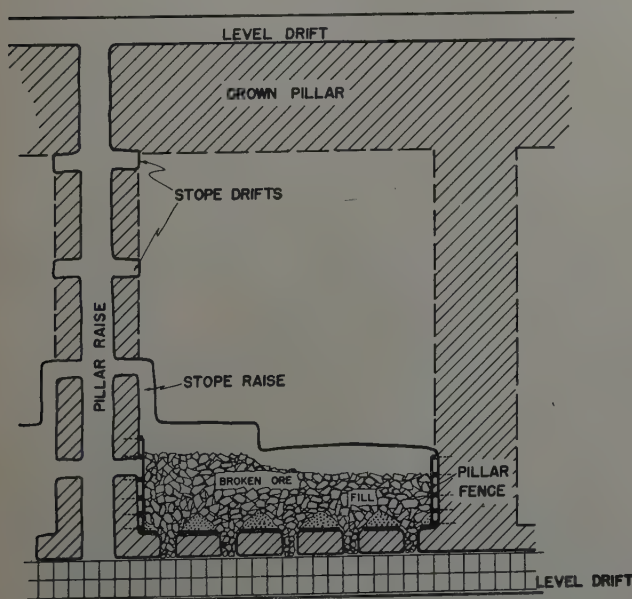


Fig. 14—Section of longitudinal shrinkage stope.

BLAST HOLE STOPING

Fig. 15—Sublevel blasthole layout, Froid-Stobie No. 7.

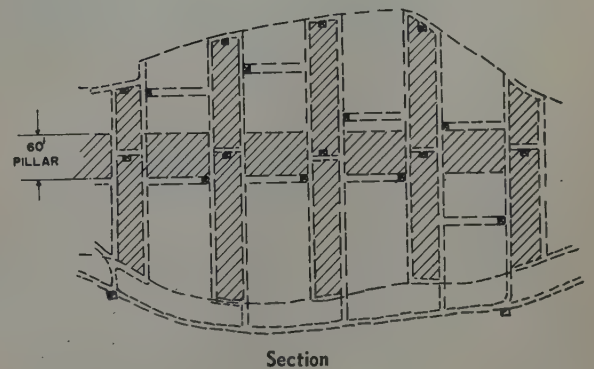
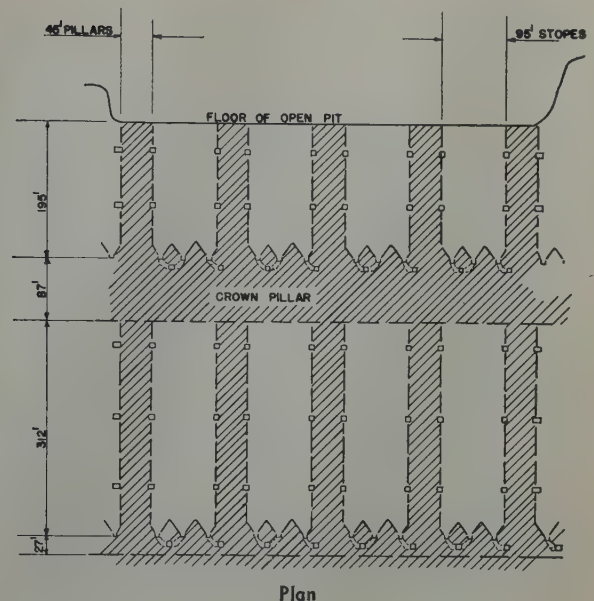


Fig. 16—Sublevel blasthole stopping layout, Froid-Stobie No. 3.

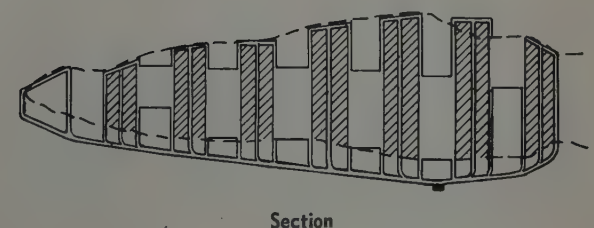
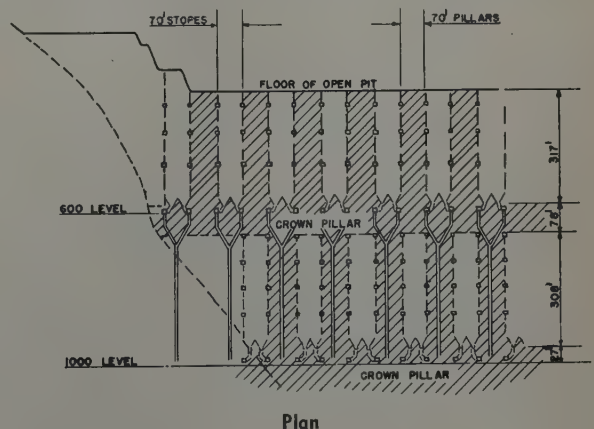




Fig. 17—Blasthole stope breaks through to open pit.

ground and the sand fill plant at surface for control of sand delivery.

Sand for filling at Creighton is recovered from tailing at the new Creighton concentrator. The tailing is deslimed and the sand is piped underground. The distance the sand is piped to reach stopes on the deepest level totals 9500 ft; more than half this distance is horizontal.

In preparation for filling, the manway and the concrete chutes are raised one floor, and repairs are made to the chutes if required. Rails in the bottom of the slusherways, floor plank, and grizzly rails are all taken up and installed on the floor above. Pillar fences on the slushing floor are completed and the manway set, the set over each concrete chute, and the chute wings are enclosed with burlap.

Panels of horizontally slotted 2-in. plank are built into one of the chutes to allow the water to run off from the sand fill. The water decants through the openings in the panels and drains through the chute to the level ditches.

Fill pours, see Fig. 8, are made alternately on the hanging wall and footwall sides of the decant chute, starting at the stope limits and working toward the chute. Each pour is controlled to produce a slope down to the decant point, and when the sand has risen in the stope to the point where it begins to run through the decant panel opening in use, that opening is closed and the water decants through the opening next above it. The average rate of placing sand fill in a stope is 250 tons per hr.

Square-Set Pillar Mining

Rib Pillars: The four-set transverse pillars in waste or gravel-filled stoping areas are recovered by a combination of overhand and underhand stoping methods. The pillars are each divided into two equal longitudinal slices. The first slice is

mined by standard overhand square-set stoping and the second by underhand stoping. Recovery is started in alternate pillars and each is mined out before work is started in either of the adjoining pillars. The overhand slice is normally stoped the full length and height of the pillar, while the underhand slice is divided into four tiers of stoping blocks, each block five to six sets long and six to eight sets deep.

When the overhand slice has been stoped out and filled, stoping is begun in the upper hanging wall block of the underhand slice and blocks against the hanging wall are mined in succession from the top to the bottom of the slice. This order of mining is repeated in each succeeding vertical row of blocks working toward the footwall until the underhand slice has been completely mined.

The method of silling the overhand slice in a pillar stope is the same as the method of silling with drifters in a regular square-set stope except that no pilot drift is required. Beginning at one of the extraction drifts the sill breasts are slashed to the free face of the pillar and the broken ore is slushed to cars in the extraction drift. The slushing equipment used for this work is a 7½-hp double-drum, air-operated slusher hoist with 36-in. scraper, standard for all pillar slushing.

The overhand slice is timbered with standard square sets, matched to those in the adjoining stopes to make the best possible fit for the spreaders used in the underhand slice. Before the first fill is placed in the overhand stope a standard sill floor is laid, chutes are established, the gangway in the extraction drift is sheeted and lagged, and the pillar fence is built as in the regular square-set stopes.

The overhand slice, see Fig. 9, is mined in a series of 7-ft horizontal floors. Each floor is opened at the fill pass manway by the mining of a slot 1 set wide from the manway across the width of the slice, and stoping is advanced from this slot to the limits of the orebody in a series of breasts 1 set long and 2 sets wide.

Drilling, blasting, slushing, timbering, and filling practices are the same in overhand pillar stopes as in regular square-set stopes.

In the underhand slice the upper stoping blocks are opened from the travelway under the crown pillar, and stoping is started at the chute. The block is mined as an underhand rill inclined at 52°. Each new floor is opened at the chute and the ore

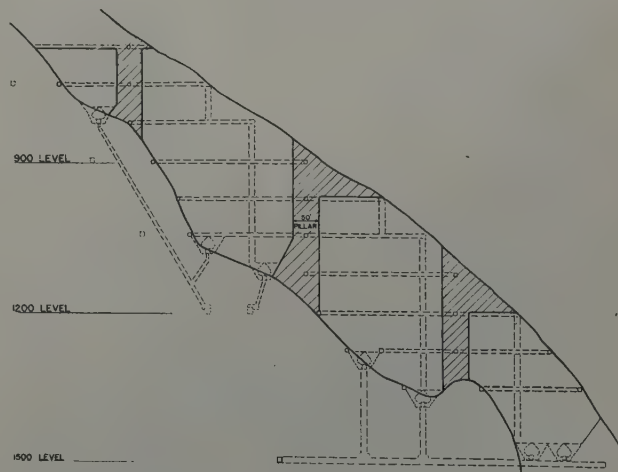


Fig. 18—Section through panel of blasthole stopes, Murray mine.

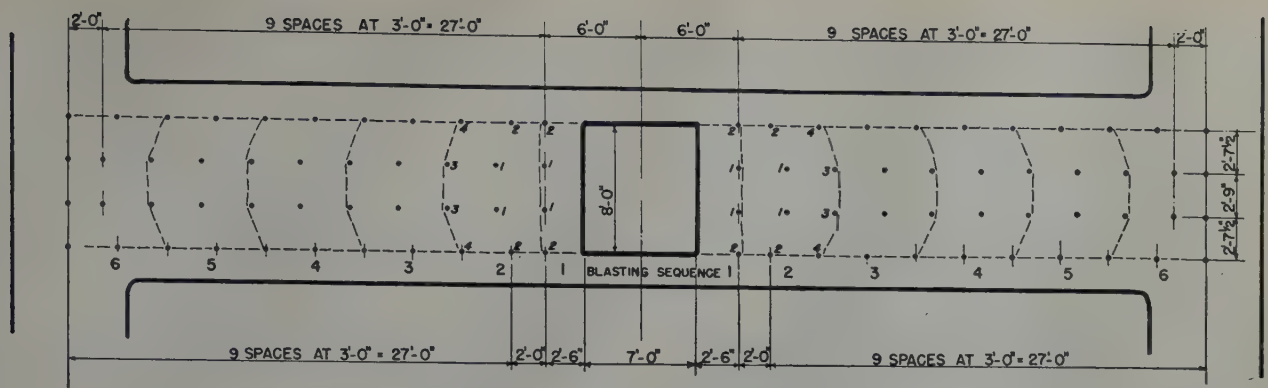


Fig. 19—Pattern of blastholes for opening stope slot.

is stoped down the rill in a succession of benches. Bench drilling is done with 55-lb pluggers, detachable bits, and drill rods up to a maximum of 7 ft long. Top floor widening is done with drifters.

The underhand stoping blocks are timbered with spreaders to keep the fences on each side from collapsing, and the spreaders are reinforced with A-braces on the two top floors of each block where the stope timber must also support either the ground in the crown pillar or the floor of a block that has been filled.

The spreaders against the ground and those that support the fill floors are 10x10-in. fir; the rest of the spreaders and the A-braces are 8-in. flatted pine. All spreaders are 11 to 13 ft long with plain ends. They are spaced 5½ ft apart horizontally and 7 ft vertically and are supported at each end on the pillar fence posts. Corresponding joints in the posts on each side of the underhand stope are cut out to provide a 4-in. bearing for the spreaders, and 3-in. plank cleats are nailed to the face of the posts for added bearing where required.

As the top floor is opened the back spreaders of 10x10-in. fir are put in place under the crown pillar. They are blocked to the ground with 8-in. bulkhead timber and are lagged with 3-in. plank. As in all stopes, ground above the spreaders is supported on booms while the spreader is being erected. The spreaders in the rows below are installed and A-braces erected as soon as room is made for them. A floor of 3-in. plank is laid from spreader to spreader at each 7-ft elevation, the floor plank secured by nailing to the spreaders.

The underhand slice of a pillar has usually been crushed to such an extent that much of it is loose which can be removed by barring. Before drilling is started on a bench all the loose is barred off and the bench is carefully examined for missed or cut-off holes. Standard practice calls for holes drilled vertically to a depth of 7 ft, but on account of the fractured ground many cannot be drilled to the desired depth and the average depth is nearer 5 ft. Holes are loaded for blasting according to the standard practice in regular stopes. Powder averages three sticks per hole. Each blast is limited to the number of holes required to make room for one spreader.

Except for the last four floors at the bottom of each stoping block the ore in the underhand pillar stopes runs down the rill into the chute as it is broken. When the rill on these floors is being levelled the broken ore is slushed to the chute with a slusher set up on the bottom floor.

To prepare a block for filling, the chutes and manways beside the block and the chute and manway to be left open in the block are lagged with 2-in. slabs. A fence of 2-in. slabs is built from top to bottom of the block on the second vertical line of spreaders one set from the footwall side, leaving a vertical row of sets open to be converted into a chute and manway for the next block. In the bottom row of blocks a sill floor is laid at the same elevation as the sill floor in the stopes on each side.

In stopes to be filled with waste rock or gravel, no further preparation is required before placing the fill, which is transferred from the stope fill raise either directly by slides or by tramping. For the top blocks the fill is dumped from the travelway at the top of the overhand slice, and for the lower blocks it is passed through slides or is trammed to the ore chutes and diverted into the blocks by bulkheads in the chutes. Floor plank, spreaders, and A-braces are recovered as filling progresses. Top floor openings, no longer required, are back-filled.

Underhand slices to be filled with water-borne sand are divided into blocks the same as those to be filled with waste rock.

When the overhand slice has been stoped out and filled, stoping is started in the bottom hanging wall

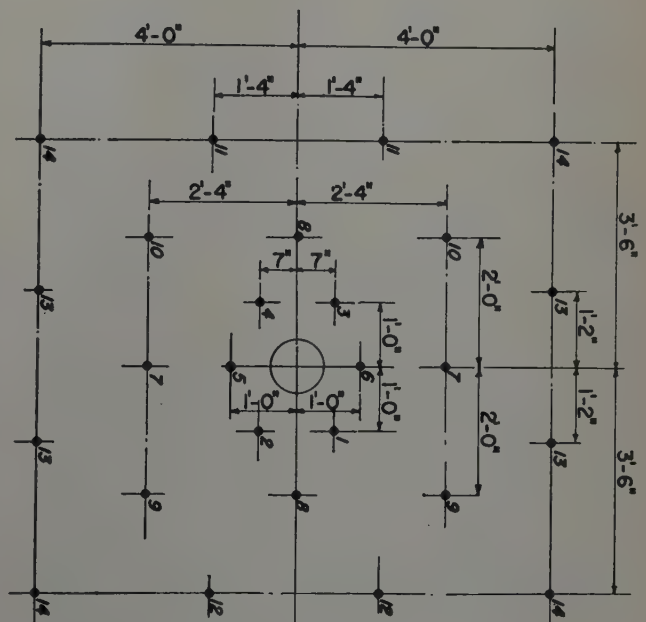


Fig. 20—Drill hole pattern in diamond drill slot raise.

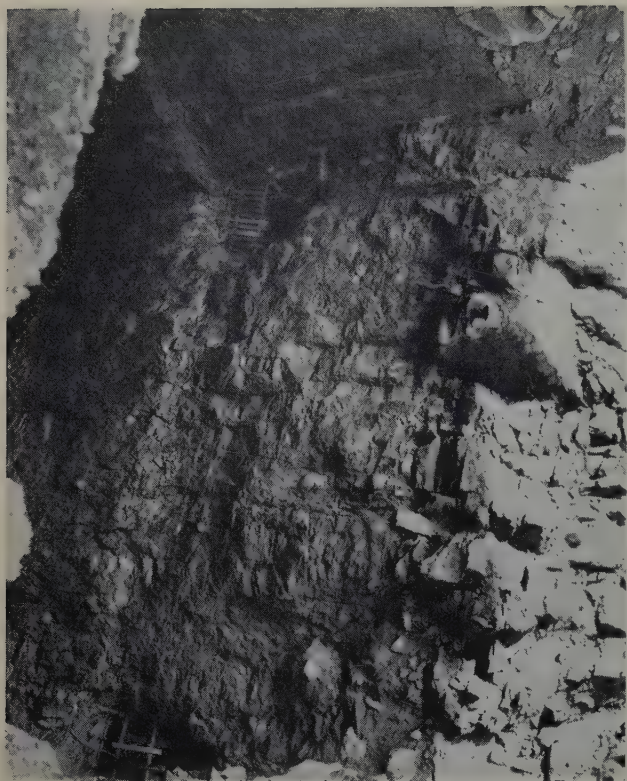


Fig. 21—Stope slot opened by churn drilling.

block of the underhand slice and the blocks against the hanging wall are mined in succession from the bottom to the top of the pillar. This sequence of mining is repeated in each succeeding vertical row of blocks working toward the footwall. When the bottom block of each row has been mined, a standard sill floor is laid, the set to be kept open is lagged, and the block is filled to the back. For the middle and top blocks all that is necessary before filling is to lag the sets to be kept open for mining the adjoining block.

Mining Pillars in Sand Filled Stopping Areas

Rib Pillars: Recovering four-set pillars between stopes that have been sand filled or sand packed is a much simpler operation than mining those between waste-filled stopes. Sand filling and sand packing the stopes has been found to stabilize the remaining ground to such an extent that the pillars can be mined overhand in the same manner as ordinary stopes. Starting from the extraction drift gangways, the sill and second floor in the pillar are stoped out over the full pillar area, the sill is filled, and standard 7-ft horizontal floors are stoped from the pillar in regular succession.

Standard concrete chutes are used, following the same general layout as in ordinary stopes. Sand filling also is done in the same way as in ordinary stopes, and the preparations for filling are the same except that no fences are required, since the pillar stope fill is contained by the sand in the stopes on each side and the wall rocks at each end of the pillar stope.

Crown Pillars: Crown pillars are mined overhand in three-set transverse slices that are mined in three groups, the working stopes in each group being separated by two dormant stopes. A slice may be mined as one stope, or in wide sections of the

orebody it may be divided into two or more stopes. In heavy ground the stope length may be as little as four sets.

Broken ore from the first two floors of a stope in the crown pillar is slushed over the fill in the stope below the crown into the boxhole. The slusher hoist is set up in a station cut in rock on the footwall side of the boxhole. When the first two floors have been stoped out a chute is timbered from the boxhole to the top of the first floor and the floor is filled. Broken ore from the remaining two floors is slushed over the first floor fill into the chute. On the sublevel the broken ore is slushed from the boxholes to the ore pass raise with a 15-hp slusher hoist and 42-in. scraper.

For filling stopes in crown and floor pillars sand pipelines are brought down into the floor pillar from the level above or up from the level below, whichever happens to be the most convenient; in crown pillar stopes the sand pipelines are always brought in from the level above. Before filling is started the pillar fence and gangway lagging is covered with burlap and arrangements are made to decant the water into an open set or a chute. No sill floor is laid in crown or floor pillar stopes. In filling the first floor of a stope the sand is allowed to permeate the fill below the floor pillar, further to stabilize the ground in the immediate area.

At mines where sand fill is not available and waste rock or gravel is used for filling stopes, the crown pillars and floor pillars are mined in a combination of alternate two-set overhand and underhand slices. Fill for floor pillars is supplied from the level above through raises in the footwall and is trammed to the stopes in 16 cu ft end-dump cars.

Cut-and-Fill

Transverse and longitudinal cut-and-fill stoping methods are both used at Inco, the choice between



Fig. 22—Blasthole drilling with tungsten carbide bits.

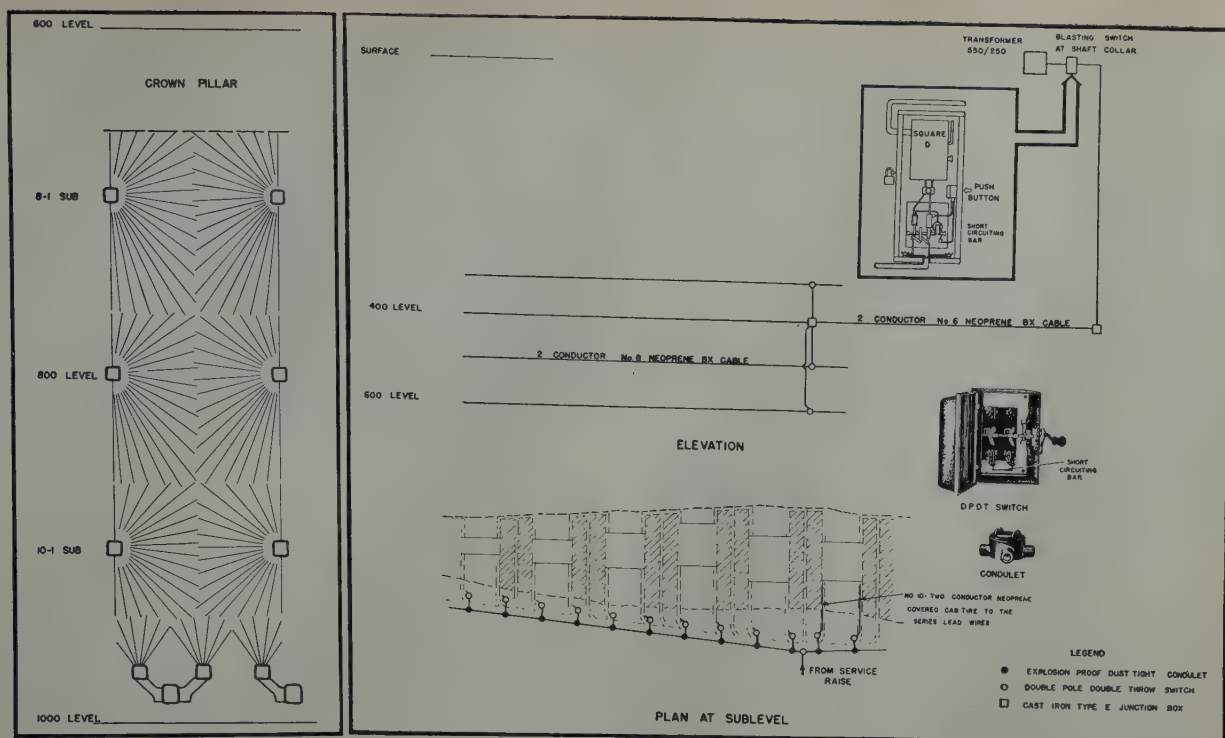


Fig. 23—Blasthole pattern for drilling with tungsten carbide bits. Fig. 24—General arrangement of permanent blasting circuit.

the two methods being determined by the width of the orebody to be mined. Orebodies less than 40 ft wide are mined by longitudinal cut-and-fill stoping. Parts of an orebody that swell to more than 40 ft in width are mined by transverse cut-and-fill stoping.

For longitudinal stoping, and for transverse stoping where an orebody is not more than 100 ft wide, only one extraction drift is needed. Two extraction drifts at approximately 50-ft centers are required for transverse stoping where an orebody is more than 100 ft wide.

Horizontal cut-and-fill stoping varies from the standard square-set-and-fill method only in the changes in technique made possible by mining without sets; deeper breast holes are drilled and more ore is broken per blast; slushing is simplified and overall stoping efficiency is higher. The general layout for a stoping block is identical, as are stope and pillar widths and the layout of chutes and manways. A stope or block of stopes can be readily converted from cut-and-fill to square-set stoping if a change in ground conditions should make the changeover advisable.

The permanent sill floors are the same as in square-set stopes. Pillar fences are built of 5-in. flatted pine posts 8 ft long and slabs 11 ft long. The posts are placed at 2-ft 9-in. intervals; each post overlaps a post in the fence on the floor below by 1 ft and is inclined its own thickness toward the wall to keep the general line of the fence in the vertical plane. The posts are blocked at the top and center to hold them in position and are sheeted with 11-ft slabs. Five-inch flatted pine timbers 11 ft long are used for laying the slushing floors. When mining of a slice has been completed this timber is taken up and relaid on the floor above.

At Garson mine, where stope fill is run of pit gravel, concrete is being used for slushing floors in approximately half the cut-and-fill stopes. In these stopes a 4-in. floor, see Fig. 10, is poured on the fill.

The concrete is mixed in the stope in 3-cu ft air-operated mixers using fill gravel and cement in the ratio of 5 to 1, and is poured as soon as possible after the fill is placed to allow plenty of time for the floor to harden before ore is blasted down on it.

Chutes in cut-and-fill stopes are concrete and are constructed in the same manner and dimensions as those in square-set stopes. On the first ten floors in the stope the chute bottom is made 20 in. thick and the sides and top 12 in. thick. Above the tenth floor the wall thickness is decreased, the bottom being 18 in. thick and the sides and top 10 in. When it becomes necessary to change the inclination of the chute, the thickness of the bottom and top walls is increased to 24 and 36 in. respectively for a distance of 14 ft below the point of change. No wings are built and no grizzlies are used on the chutes. Chute openings at the stope slushing floor are guarded by pipe railings suspended from the back of the stope.

Blasting chambers for use when a *hang-up* occurs are built from each chute to the nearest manway at 5-floor intervals and an opening 20 in. square is left through the wall of the chute at each blasting chamber. The chambers are 3 ft wide and 5 ft 9 in. high inside and are built of special sets framed from 8-in. flatted pine; the sets are lagged on top with 5-in. flatted timber and are sheeted with pine slabs.

Development and Stoping: Standard transverse cut-and-fill stopes are 30 ft wide and pillars are 4 sets wide. There are two manways in a stope, one against each pillar, and one or two chutes, depending upon the number of extraction drifts.

Standard longitudinal cut-and-fill stopes, see Fig. 11, are 200 ft long. Four-set pillars are left between stopes in the wider sections of the orebody, but in sections narrower than 20 ft the stopes are joined together. There are three chutes in each stope, spaced at 58-ft centers, and three manways, one at each end and one near the center of the stope.

All cut-and-fill stopes are silled 30 ft above the base of rail.

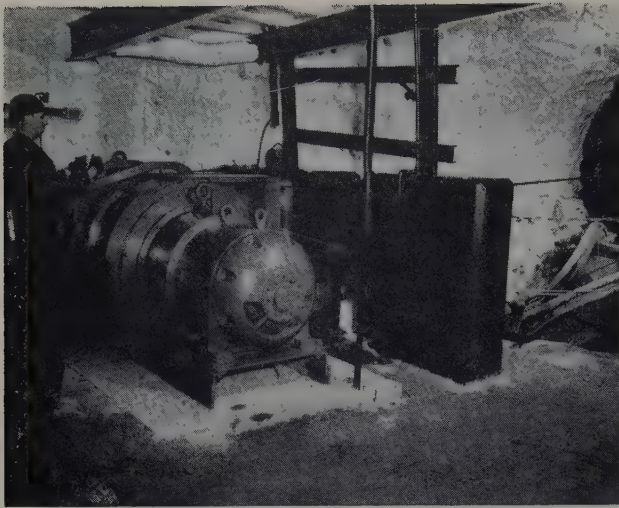


Fig. 25 (left)—A 125-hp slusher hoist slushing to ore pass.



Fig. 27 (right)—A cantilever tail sheave.

Extraction drifts are driven in ore and are timbered with standard two-floor gangway sets. The boxholes are 5 ft square and are driven from the upper floor of the gangway to the stope sill. Openings for the manway bases are made in the side of the extraction drift before the drift is timbered. The manway raises driven from these openings to the sill floor are 7 ft square and are timbered with standard pine cribbing. One 7x11-ft fill raise with a cribbed manway compartment and an open compartment for fill is driven in each stope. In a longitudinal stope the raise is driven in ore on the footwall contact and approximately in the center of the stope along the strike. In a transverse stope the raise is near the footwall beside a pillar.

Detailed exploration in narrow orebodies is done by diamond drilling from a 5x7-ft exploration drift driven along the strike of the orebody at the stope sill floor elevation, either in ore or in the hanging wall rock. When the drift is in ore short connections are driven to the tops of the boxholes so it can be utilized as a pilot drift for stope silling; otherwise a pilot drift is driven at the sill floor elevation from the top of one boxhole to the next.

The sill of the stope is cut 7 ft high and silling is

done by slashing to the pilot drift and slushing the broken ore into the nearest boxhole. When the silling reaches the stope fill raise position the raise is begun and is completed by the time the balance of the sill and the second cut in the stope have been mined. Laying the permanent sill floor, building pillar fences, and raising the chutes and manways in preparation for filling the sill floor follows the mining of the second cut.

The slices mined from the back of the stope are 7 ft high. A new slice is started at the fill raise by mining a slot 14 ft wide across the width of the stope, and succeeding blasts are drilled from setups on the broken ore. Drilling, see Fig. 12, is done with 2½-in. piston diameter air-leg drills, and ⅞-in. hexagonal drill steel. The bits are forged integral with the steel and are chisel-type with tungsten carbide inserts. Drill steel is made up in three lengths, 5.25 ft, 10.5 ft, and 15.75 ft. Bit gages are 1.30 in., 1.22 in., and 1.14 in. respectively.

Holes are drilled to break a slice of ore the width of the stope and 15 ft long. The back holes are drilled to form a smooth arch, the holes in the center of the stope being drilled higher than those at the walls. The holes in the rows below are staggered,

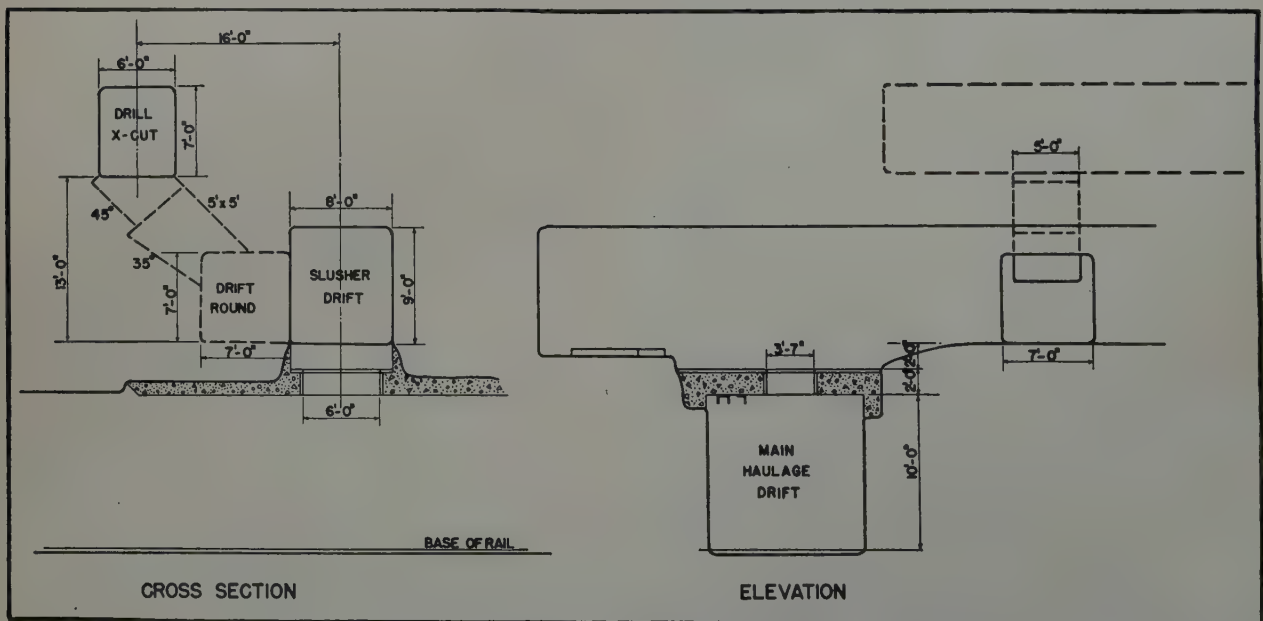


Fig. 26—Section through a slusher drift showing connection to haulage drift.

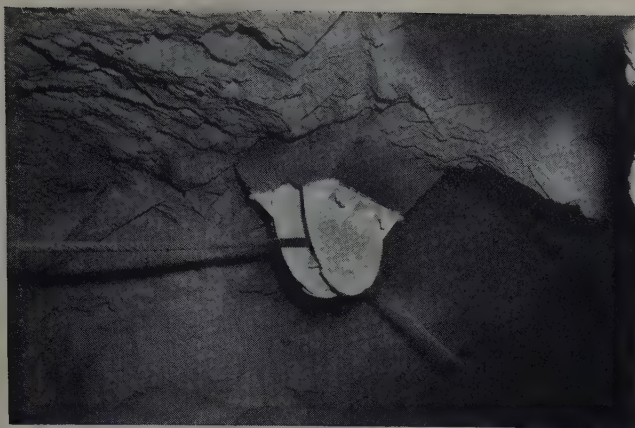
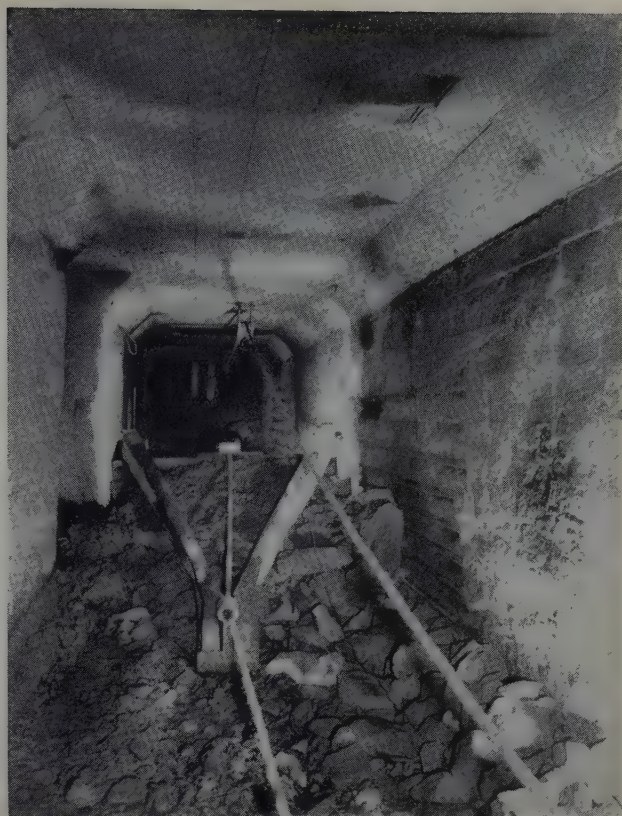


Fig. 28 (left)—A tail rope idler sheave. Fig. 29 (right)—A concreted slusher drift with 72-in. folding scraper.



but the number to be drilled and the spacing depends on the shape of the breast and overbreak from the previous blast.

For primary blasting, and for secondary breaking when the blockholes and sandblasts are fired with the primary blast, standard explosive is used with short period delay electric blasting caps. For blockholes and sandblasts to be fired independently of a primary blast, 5-ft. tape fuse and No. 6 blasting caps are used instead.

With the short period delay caps, less loose ground develops, hanging wall overbreak is reduced, and better fragmentation is obtained, resulting in lower secondary powder consumption.

Broken ore is slushed to the chutes with a 15-hp air-operated slusher hoist and a 42-in. scraper shown in Fig. 13. Both solid and folding-type cast manganese steel scrapers are used and the cables for both the haul and pullback lines are ½-in. diam.

The slusher is anchored to a heavy timber buried in the fill or by chains to the walls of the stope. As a safety measure for the protection of the slusher-man a standard square set is erected around the slusher and a crib of 8-in. flatted timber is built up from the top of the square set to the back of the stope. The broken ore, except for that needed for the drill setups, is removed after each blast.

Fill is run into the stope from the fill raise and is levelled and spread with a slusher for a distance of approximately 50 ft each way from the fill raise. A simple chute is then built in the raise and the balance of the fill is spread with a standard fill car. Track for the fill car is laid with 16-lb rails in 11-ft lengths held together by removable tie plates. Chutes and manways are raised in advance of the filling.

Shrinkage Stoping

In the early days at Inco the major part of the ore produced from underground was mined by the shrinkage stoping method. This was the principal method used at Creighton in mining down to the 1600-ft level below surface, at Levack to the 750-ft level, and at Garson to the 1200-ft level. The method is still in use at Garson mine, and shrinkage stopes have been opened as far down as the 2000-ft level in places where the orebody is relatively narrow. In the wide orebodies at Levack the shrinkage method has given way to blasthole stoping.

Many modifications of the method have been developed and used. The earliest variation at Inco was shrinkage stoping with round pillars and dry-wall drifts. These stopes were silled at the floor of the

level to the limits of the orebody, and dry-wall tramping drifts, 7x7 ft inside with chutes spaced at regular intervals, were built as the silling advanced. With the advent of power tramping, extraction drifts with boxholes were driven mainly in ore, and the stopes were silled 25 ft above the level. Round pillars were left in the stopes to support the back, and the stope covered an entire level.

The next development in shrinkage stoping was to reduce the size of the stopes and use rib pillars. In wide orebodies transverse stopes with rib pillars became the standard. The width of the stopes and pillars varied at the different mines from 50-ft stopes and 25-ft pillars to 100-ft stopes and 100-ft pillars. In narrow orebodies longitudinal shrinkage stoping was the practice, but there was no set length for a longitudinal stope, the pillars being established where there was a pinch in the orebody or where the grade of the ore was lower than average.

The present standard in open stoping areas is to mine an orebody less than 50 ft wide by longitudinal shrinkage stoping with subsequent fill, and an orebody more than 50 ft wide by blasthole stoping.

Longitudinal Shrinkage Stoping: Standard longitudinal shrinkage stopes are 130 ft long, measured along the strike of the orebody, and are mined from a 24-ft sill to within 40 ft of the level above. Pillars 30 ft wide are left between the stopes.

Stopes are developed by a 10x10-ft extraction drift driven in ore along the strike in such a location that boxholes opened from the drift will give good coverage of the stoping area at the sill floor of the stope. The drift is driven without timber and is slashed to a finished height of 16½ ft to make room for standard gangway sets.

Boxholes 6x6 ft at 27½-ft centers are driven from one or if necessary from both sides of the extraction drift from the upper gangway sets to the sill floor of the stope. The loading chute is installed after the second round in the boxhole has been

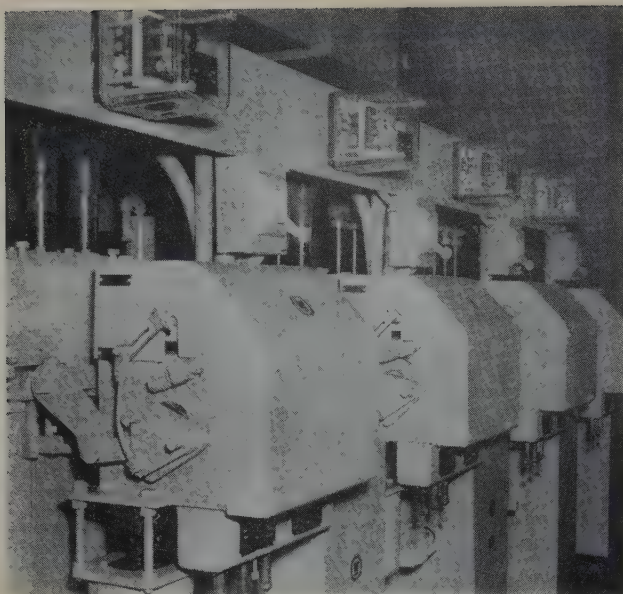


Fig. 30—Circuit breakers in slusher hoist control room.

blasted. The sides of the chute are 10x10-in. timber and the bottoms, built at a slope of 35°, are 6x8-in. timber covered with ½-in. steel plate. The control gate is a cast-steel underslung arc gate operated by an 8-in. diam compressed air cylinder. Supplementing the gate is an 8-in. round stop log suspended horizontally above the gate by chains from the gangway posts.

A 7x11-ft service raise is driven in ore close to the footwall from level to level in each pillar, and as the raise is advanced 4x6-ft connection drifts are driven at 35-ft vertical intervals from the raise into the stoping areas on each side. The raise contains a cribbed manway compartment with ladderway and timber slide, and an open compartment which handles broken ore during the driving and serves as an airway while stoping is in progress. The raises are subsequently stripped and used as passes for ore from crown pillar mining.

Stoping: The stopes, see Fig. 14, are silled at an elevation 24 ft above the base of rail of the level. Two boxholes are driven to the sill floor of the stope and are joined by a 5x7-ft pilot drift. One of the boxholes is used for broken ore disposal during the initial stage of silling and the other serves as a manway while silling is in progress. The rest of the boxholes are driven one at a time as they are needed to keep pace with the silling.

When silling has been completed to one of the pillars a 5x5-ft manway raise is driven from the back of the stope to the first connection drift from the service raise in the pillar. The manway provides access to the stope and is the starting point for mining each slice above the sill.

The second floor is mined in exactly the same manner as in a cut-and-fill stope, and as the broken ore is removed standard sill floor is laid between boxholes. Pillar fences are also built, but are of heavier material than in cut-and-fill stopes and are bolted to the ground to hold them in place when the broken ore is being drawn from the stope.

The fence consists of 8-ft vertical panels of 5-in. flatted jackpine spiked to horizontal bearers of 5x8-in. x 8-ft fir. The bearers are supported and held

in place by 1¼-in. diam hooked rock-bolts set 5 to 7 ft into the pillar and spaced at 4-ft centers.

To protect the pillar fences the boxhole at each end of the stope is driven in the second set from the pillar so that as the ore is drawn it will roll toward the center of the boxhole instead of sliding down along the face of the fence.

The sill floor of the stope is filled with waste or gravel to a depth of 8 ft, and upon completion of filling, excess fill is drawn from the stope until cones are formed around the chutes at the angle of repose of the fill. Fill is supplied by way of the open compartment of the pillar raise and the stope manway raise to the stope and spread by slushing.

Shrinkage stopes are mined in a series of horizontal slices, each slice started by opening a cut 14 ft wide across the full width of the stope at the manway raise. Drilling equipment is the same as in cut-and-fill stopes. Holes are drilled 13 to 15 ft deep and are spaced 6 ft apart horizontally and 2 ft apart vertically. They are loaded with standard explosives and detonated with short period delay electric caps.

Slices 8 ft in height are mined until the stope is high enough for the broken ore to settle evenly as the boxholes are drawn. Subsequent slices are mined 10 ft high. Forty per cent of the ore broken is immediately available. As stoping proceeds, broken ore is drawn down ahead of the face to 10 ft below the back to make room for the drill setups and the ore broken in the next blast. As mining approaches the top of the first manway raise, the raise is extended 35 ft to the next manway drift above. When a stope has been mined to the base of the crown pillar, a 5x5-ft fill raise is driven to the extraction drift on the level above and the broken ore is drawn from the completed stope.

After the shrinkage ore has been drawn, waste rock or gravel is dumped into the stope through the fill raise. In the final stage of filling, the fill is levelled off 10 ft below the crown pillar by slushing.

The crown pillar is divided into a series of transverse four-set blocks for recovery by square-set mining. A slusherway is established on the fill from one pillar to the other, and the broken ore from the crown pillar is slushed to one or the other of the two service raises, which are stripped to serve as ore passes.

Rib pillars are mined by the standard square-set pillar recovery method after the crown pillars on each side have been mined out.

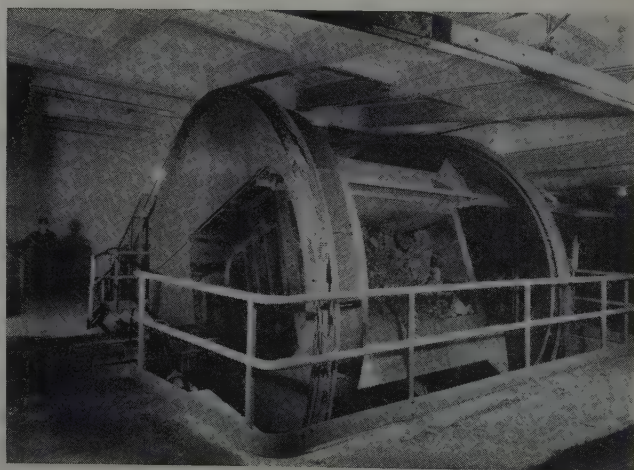


Fig. 31—Rotary tippie dumping 260 cu ft cars.

Sublevel Blasthole Stoping

Sublevel blasthole stoping is being used in open stoping areas for mining low and medium grade orebodies more than 50 ft wide. Used at Levack since 1939 and experimentally in recent years at Creighton the blasthole method has been adapted for mining at the Murray and Frood-Stobie mines. The method has become of major importance in conversion to all underground production, and on completion of the program more than 50 pct of the ore produced will be taken from blasthole stopes.

General Layout

Frood-Stobie Mine, Stobie Section: At Frood-Stobie blasthole stoping is in progress in two separate mining areas, one in the Stobie section and one in the Frood section. The Stobie section is being developed and the new mining plant is being constructed for a daily production of 18,000 tons.

The mine is being developed through No. 7 shaft, the main hoisting and supply shaft, containing two skip compartments, two cage compartments, a pipe compartment, and a manway compartment. A second shaft, No. 8, for ore hoisting only, is being sunk initially to a depth of 1900 ft. No. 8 shaft will be served by a 14-ft 110-in. parallel double-drum geared hoist driven by two 3000-hp, 600-v, 500-rpm, dc motors with rotating-type control. Bottom dump skips of 16-ton capacity will be used and hoisting ropes are to be 2 1/16-in. flattened strand with a breaking strain of 430,000 lb. Hoist operation will be automatic with push button control at the loading pocket as at Creighton No. 7 shaft. Hoisting capacity will be 750 tons per hr. For flexibility of operation both on surface and underground the new shaft has been located within 100 ft of the No. 7 shaft. Underground conveyors carry the ore from the main ore pass system to a loading station at each shaft and the tonnage delivered can be varied in any proportion desired.

On surface, the ore is treated in a common crushing plant and can be conveyed from the receiving bin to either or both of two secondary crushers.

The mine has been opened from levels at 200-ft intervals, and the 600, 1000, and 1400-ft. levels are main haulage levels. Ore is trammed on each of these levels to a crusher station where it is crushed to 6 in. in a 66x48-in. jaw crusher. For each crusher station there is a 2500-ton storage bin. From the

crushers the ore falls to the bins and is conveyed as required to one or the other of duplicate systems of vertical passes leading to the loading station conveyor level. To minimize wear the ore passes are kept full, the ore being delivered to them at the same rate it is withdrawn.

The initial stoping block, see Fig. 15, lies beneath the Stobie open pit, where mining has been completed, and extends from the pit bottom to the 1400 level. The orebody in this area varies in width from 250 to 400 ft and dips at 75°. The length of the block is 1000 ft.

Because of its width the block has been divided longitudinally by a 60-ft pillar into two series of 95-ft transverse stopes and 45-ft pillars. The longitudinal pillar is offset on each main level to make the tonnage in the stopes on each side approximately equal. The center lines of the transverse pillars coincide on all levels.

Stopes are being mined on the three main levels. On 600 level they are being broken through to the open pit, and are 195 ft high. On the 1000 and 1400 levels they are being mined to the base of a crown pillar 87 ft from the level above and the height from sill to crown is 312 ft.

The broken ore is slushed into cars on the main haulage levels from slusher drifts, 14 ft above base of rail, to a hanging wall extraction drift from one series of stopes and to a footwall drift from the other. The maximum slushing distance is 175 ft.

Slusher drifts are established at 72½-ft centers, one on the edge of the pillar, the other under the stope. This distance is equivalent to five car-lengths for simultaneous loading from adjacent drifts. For regular stoping, three rows of boxholes spaced at 25-ft centers provide stope coverage. In the drift under the stope boxholes are driven on both sides; in the other drift boxholes are established on the stope side only, those on the opposite side being



Fig. 32—A 20-ton locomotive and train of 260 cu ft tipples cars.

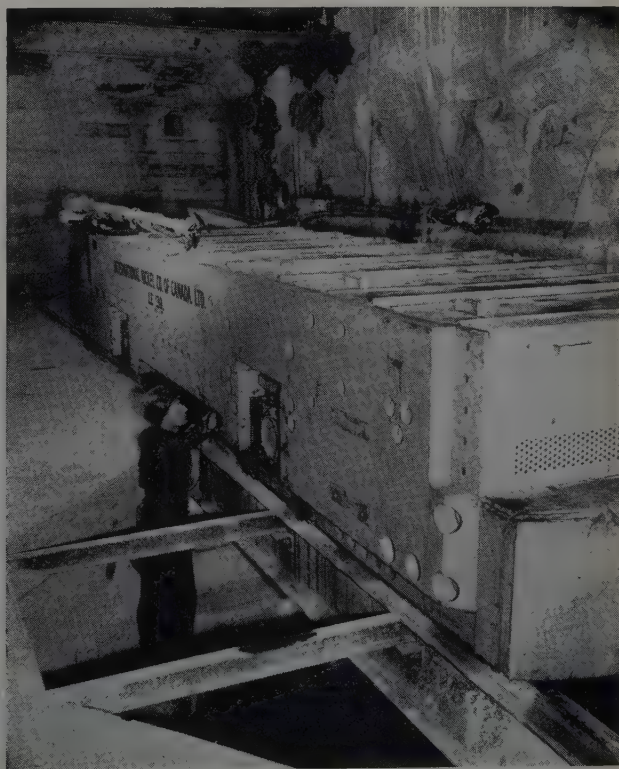


Fig. 33—A 20-ton locomotive in underground repair station.

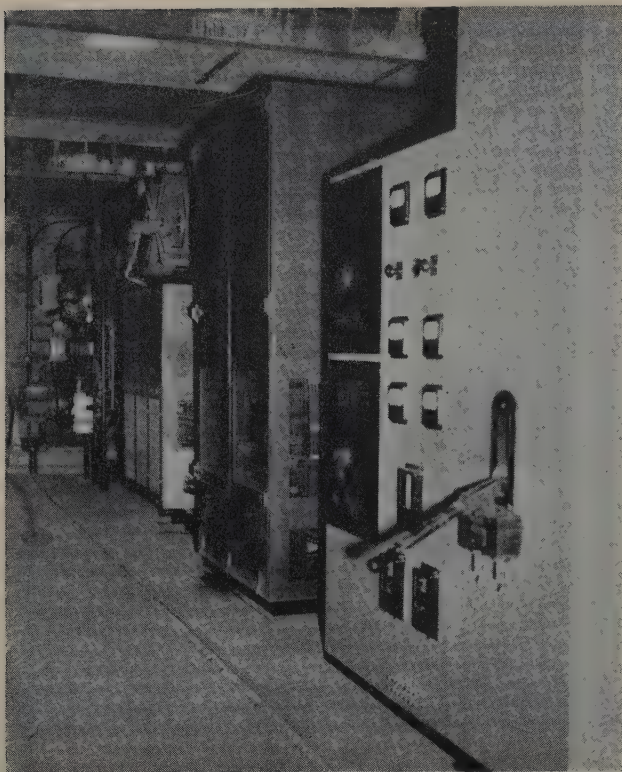


Fig. 34—Underground rectifier station.

driven later for pillar recovery. Sill crosscuts are driven from footwall to hanging wall 14 ft above the floor of the slusher drift on the boxhole center lines, and slot raises from the sill crosscuts connect with the sublevel development.

Service raises, from which the sublevels are established, provide safe access to all stoping faces. They are driven from the main haulage levels, two in each rib pillar, one at the hanging wall, and one at the center of the longitudinal pillar. In addition a raise is driven in rock from the footwall extraction drift at each end of the stoping section.

Sublevel development is established at 100-ft intervals from the service raises; footwall drifts are 6x7 ft in cross section while drill crosscuts located on opposite sides of each stope are 7x7 ft. Broken rock from the sublevel development is slushed to the service raises.

Frood-Stobie Mine, Frood Section: In the Frood section, the portion of the orebody being mined by blasthole stoping is 1200 ft long, 200 ft wide, and dips uniformly at 75°. The block extends from the 1000 level to the bottom of the Frood section of the open pit and has been divided on two horizons, the 600 and 1000 levels, into 70-ft transverse stopes and 70-ft rib pillars, see Fig. 16. Stoping was started on 600 level and these stopes are being broken through into the pit, see Fig. 17. A 78-ft crown pillar is being left at the top of the 1000 level stopes.

The ore on 600 level is slushed to ore passes and the ore on 1000 level is slushed direct to cars for tramming to the 1000 level crusher.

Slusher drifts on both levels are normal to the strike of the orebody and are spaced at 70-ft centers. They are driven on the lines between the stopes and pillars so that, with boxholes at 25-ft centers along one side, two drifts serve each stope. For pillar mining boxholes are driven later along the opposite sides and two drifts serve each pillar.

On 600 level the slusher drifts are driven at the level elevation from a footwall service drift and slushing is toward the footwall. On 1000 level they are 14 ft above the base of rail of the main haulage drifts and slushing is toward the hanging wall.

There are two extraction drifts on 1000 level, one in the footwall rock and one in ore near the hanging wall. Ore from the 600 level is loaded from the chutes in the footwall drift while ore from the 1000 level is handled in the hanging wall drift, to avoid interference when two trains operate in the area.

Drilling sublevels are at 70-ft intervals. Entrance to the drill crosscuts in the stopes is from footwall drifts on intermediate levels connected to the main shaft, and on sublevels from two service raises in the footwall. The service raises are 700 ft apart and extend from the 1000 level to the top of the stoping block. Pillar crosscuts with connections along the hanging wall contact to the drill crosscuts give access to the hanging wall faces of the stopes.

Murray Mine: At Murray the orebody, which is flat-dipping and 200 ft wide, has been divided into 100-ft transverse stoping panels and 75-ft rib pillars. The panels extend from 600 to 1500-ft level and each, see Fig. 18, is subdivided into three and sometimes four stopes by 50-ft vertical cross pillars from hanging wall to footwall between the ribs.

The dip of the footwall is not uniform, being as flat as 25° in places and as steep as 50° in others, with abrupt changes in dip. Advantage is taken of this characteristic in laying out the cross pillars so that they stand on the flat sections of the footwall at the bottom of the steeper stopes.

All slusher drifts are parallel to the footwall; boxhole spacing is 30 ft and maximum slushing distance is 115 ft. For the upper stopes the drifts are in rock under the orebody, one drift in each stope so located that the boxholes break through at the base of the cross pillar. Boxholes from one or more slusher drifts above this complete the stope coverage. The ore from the footwall drifts is slushed into ore passes leading to the main haulage on 1200 and 1500 levels.

For the bottom stope in each section the slusher drifts are in ore and are driven at 65-ft centers under the stope sill from haulage crosscuts in the pillars. From these drifts the ore is slushed into cars.

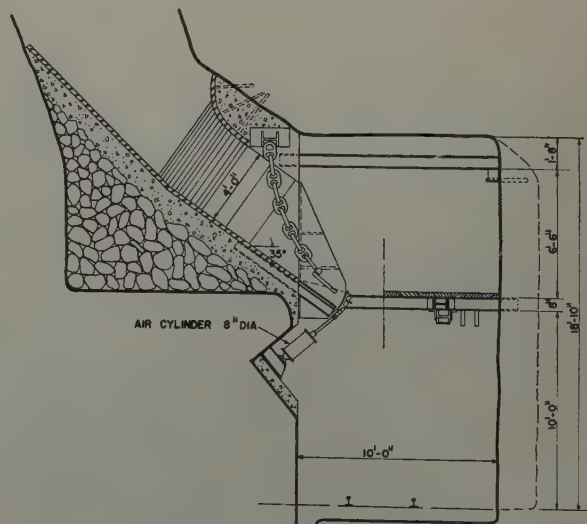
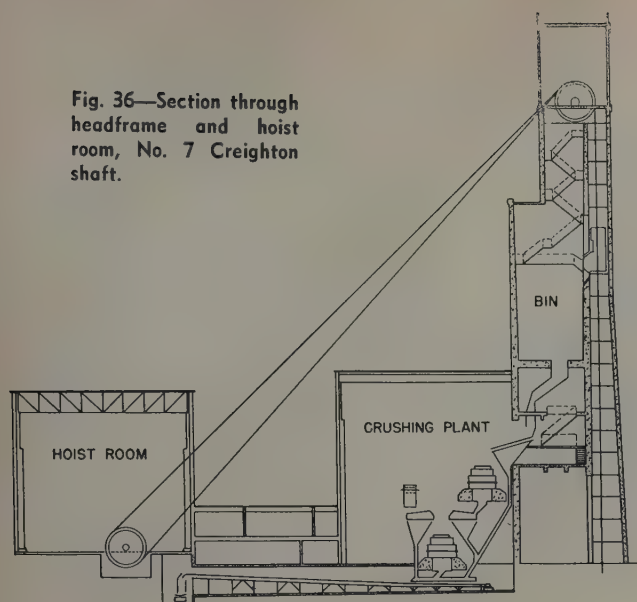


Fig. 35—Detail of ore pass loading chute for blasthole ore.

Fig. 36—Section through headframe and hoist room, No. 7 Creighton shaft.



Access to the drill drifts is from the main and intermediate levels and through service raises and drifts in the pillars.

Mining

Stopes are silled by radial drilling from 7x7-ft drifts 14 ft above the floor of the slusher drifts over the lines of boxholes. From the sill drifts radial blastholes are drilled in rings at 2½-ft centers to break a trough-shaped cut over the boxholes. The troughs are 32 ft high and 30 ft wide at the top with sides sloping at 55°, and the face is kept 25 ft ahead of the face of the stope.

The stope slot is established from a 7x11-ft raise from the sill drift to the sublevels above. Slot drifts 6x7-ft in cross section, collared at the slot raise, are driven across the stope from pillar to pillar on each sublevel and widened to 11 ft. The raise is widened to form a slot the full width and height of the stope by blasting vertical holes from the slot drift.

To break an 8-ft slot the blastholes are 2 ft 8 in. apart in rows at 3-ft centers, see Fig. 19. The holes are column-loaded with 1¼x8-in. sticks of 75 pct gelatin dynamite detonated with millisecond delay electric caps.

An alternative method of developing slot raises is being used in stopes which break through to the open pit, see Fig. 20. In this method a 9-in. churn drill hole is drilled from the pit floor to the stope sill and drifts are driven to intersect the hole on each sublevel. With the churn drill hole used as a cut, 7x8-ft raise rounds are drilled from sublevel to sublevel with diamond drills. Starting at the sill and working through to the pit floor, sections up to 25-ft in length are broken in succession from the bottom. Holes are loaded from the sublevel above.

Churn drill holes are also used for opening the section of the slot between the upper sublevel in the stope and the pit floor. Here the slot, see Fig. 21, is broken 18 ft wide across the full width of the stope in one blast. The churn drill holes are blocked 2 ft from the bottom with a concrete plug and are column-loaded to within 10 ft of the collar with 75 pct powder in 12½-lb bags, detonated with primacord and millisecond delay electric caps.

For regular stoping, blastholes are drilled from the sublevel drifts with diamond drills, or percussion drills with tungsten carbide bits. Prior to 1950,

the drilling of long blastholes was done almost entirely with diamond drills. Now that bits tipped with tungsten carbide have been developed, drilling in certain types of ground can be done with percussion drills at less cost per foot than by diamond drilling. At the Stobie section of the Froid-Stobie mine and at Levack mine nearly all blast-hole drilling is being done with percussion drills, see Fig. 22.

Blasthole Patterns: Blastholes are drilled to allow a primary factor of 0.33 lb of powder per ton of ore broken. For diamond drilling, sublevels are established at 70-ft intervals. Normally the holes are limited to a maximum of 75 ft in length, and up-holes to 15°. Toe spacing is 8 ft.

To break a 5-ft slice of ore radial blastholes are drilled in two patterns: in one the holes are all in one plane; in the other they are in two planes 2½ ft apart with alternate holes of the pattern in one of the planes and the balance in the other.

A slice 25 ft thick is broken in one blast consisting of one full ring 5 ft from the face, eight intermediate rings, and finally a full ring at 5-ft burden.

For percussion drilling, see Fig. 23, sublevel spacing is increased to 100 ft. Down-holes are limited to a maximum of 65 ft, vertical up-holes are drilled to a maximum of 35 ft, and other holes above the horizontal to a maximum of 50 ft. The pattern is similar to that for diamond drilling but the burden is increased to 6 ft to maintain the same primary powder factor.

Primary Blasting: Blastholes are loaded by the drill runners. A length of ½-in. plastic tubing fitted for air and water is used to clean the holes before loading.

The holes are loaded with 75 pct gelatin dynamite. Those drilled with diamond drills are charged with 1¼x8-in. sticks and those drilled with percussion drills are charged with 1½x8-in. sticks. Each hole is loaded solid to within 17 ft of the collar; above that for 7 ft, a 20-in. spacer is used between every three sticks, and in the next 7 ft a spacer is used between each stick of powder. The top of the hole is left open for 3 ft. The powder is tamped with a length of ¾-in. plastic tubing. The detonator is placed in the second stick from the top of the charge.

Short period delay electric blasting caps are standard for primary blasting. Caps are connected in series in parallel, each series limited to 25 caps. Each ring fires in succession, boundary holes being delayed one period behind the rest of the holes in a ring.



Fig. 37—No. 7 shaft headframe and concentrator, Creighton mine.

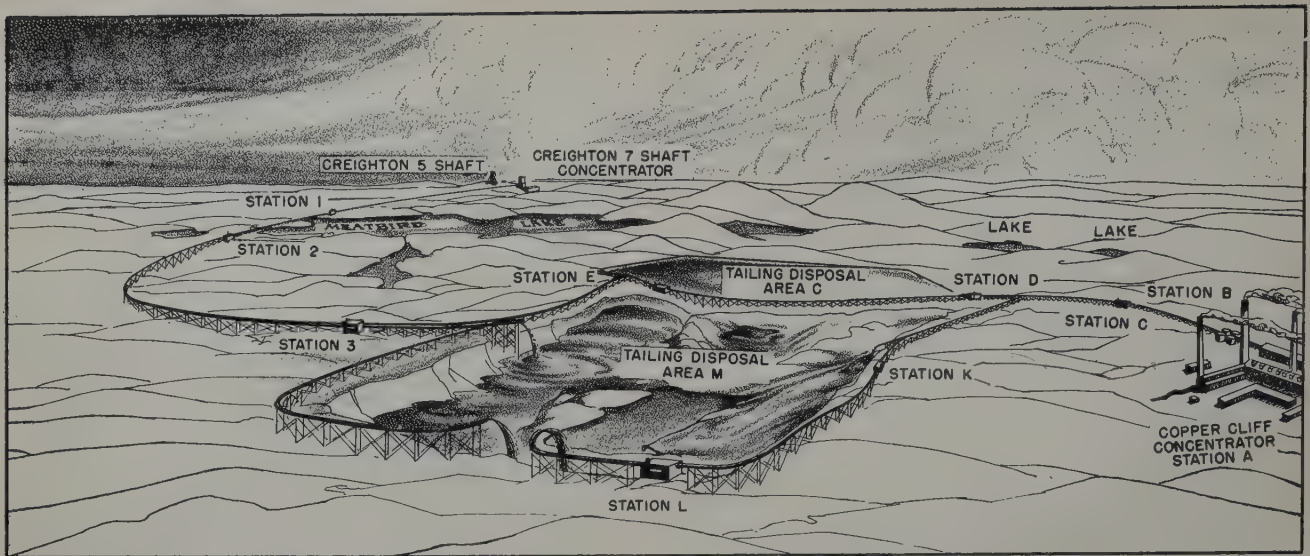


Fig. 38—General plant orientation and system for pumping concentrate and tailing.

The general arrangement of the permanent blasting lines is shown in Fig. 24. Blasts are fired from surface with 220-v ac current through a magnetic switch kept locked when not in use.

Slusher Drift Construction: Except for the discharge opening, drifts for slushing to cars or to ore passes are of the same general design. Usually ground conditions are such that no support is required, but where ground is poor slusher drifts and boxhole brows are completely concreted.

The hoist is set at a minimum of 11 ft from the discharge opening and clearance is left on all sides for operation or dismantling. The width of the hoist room is 15 ft, the length from the discharge opening is 18 ft, and the height below back steel is 7 ft 10 in.

Fig. 25 is a view of a slusher station for slushing to an ore pass. At the discharge end of the drift the hoist room and travelway are protected by an 18-in. concrete wall 3 ft high. An opening left in the travelway side, normally closed by rope guards, permits the moving of equipment in and out of the drift. The hoist is anchored to a concrete foundation with 3-ft bolts extending through the concrete into the rock. At the slusher discharge a manganese steel crusher mantle is set in concrete 2 ft below the floor of the drift. The opening through the mantle is approximately 5 ft in diam.

The general layout of the slusher drift at the discharge for slushing into cars is shown in Fig. 26. The back of the haulage drift is covered with a 24-in. concrete slab in which an opening 3 ft 7 in. x 6 ft is left for the slusher discharge. The surface of the floor slab and the sides of the discharge opening are lined with 2½-in. carbon steel billets.

A ladderway opening in the concrete floor slab provides access to the slusher drift. The slusher discharge is separated from the ladderway and the travelway by a concrete wall 18 in. thick and 3½ ft high.

Tail sheave supports of the cantilever type are installed a minimum of 16 ft beyond the last boxhole, approximately 6 ft from the end of the slusher drift. For removal of ore dragged back to the tail sheave the assembly is centered 9 ft 10 in. above the drift floor to allow the scraper to be swung over the chunks, an arrangement which also reduces

rope wear. The sheave assembly, see Fig. 27, set in a hitch and concreted in place, consists of a 20-in. diam cast steel sheave fitted with standard roller bearings and mounted on two 18-in. I-beams. The sheave, designed to pass knotted slusher rope, has a rope groove 3 in. deep, flared to 4 in.; the bottom of the groove is 11/16-in. radius for 1¼-in. rope.

Sheave assemblies consist of cast steel sheaves 6 in. wide and 6 in. in diam fitted with roller bearings and shafts mounted between two semi-circular steel plates. They are anchored with 1-in. diam bolts and concreted in place.

Assemblies are of two types; the first, with two sheaves mounted one above the other to keep the rope from contact with the back of the drift, is installed approximately 20 ft from the tail sheave; the second, with a single sheave, is installed between boxholes at 75-ft centers to carry the tail rope, see Fig. 28.

Slusher Hoists and Auxiliary Equipment: Slusher hoists, for blasthole and caving mining, are 125-hp double-drum hoists with manual lever control. They are designed for a rope speed of 275 ft per min and a rope pull of 15,000 lb and have a drum capacity of 500 ft of 1¼-in. rope. With each slusher hoist is a 3-hp tugger for handling rope and miscellaneous equipment and a sealed beam light for slusher drift illumination. Slusher scrapers are of a 72-in. folding type weighing 3½ tons and haul an average load of 2½ tons, see Fig. 29.

The electrical equipment is located in control rooms, see Fig. 30, on the main levels far enough away from blasting to be protected from concussion.

Circuit breakers are motor-operated, and four hoists are controlled from one room. Power and control cables are run through boreholes from the circuit breakers to the hoists, 3-conductor No. 6 neoprene-covered armored cable for 2200-v power and 6-conductor cable for control and lighting.

Slusher hoists are equipped with 1¼-in. diam used well-drill hoisting rope, secured to the scrapers by a hitch through the pull and tail rope brackets. Hydraulic or mechanical cable cutters are used to cut and trim broken ropes after knotting.

For stowing slusher ropes during blasting operations the tugger hoist rope, strung through eyebolts in the back of the drift and anchored near the tail

sheave, is attached to the slusher rope to hoist it to the back of the drift.

Haulage Equipment: At Frood-Stobie the new haulage equipment for handling ore from blasthole mining consists of 20-ton trolley locomotives and 260-cu ft tippie cars; at Murray the locomotives are 20-ton, but cars are the Granby type, of 188 cu ft capacity.

The locomotives each have two 120-hp, 250-v dc driving motors and are equipped with air brakes; the draw bar pull is 10,000 lb at 8 miles per hr. The tippie cars are 14 ft 6 in. from center to center of coupler, 7 ft wide overall, and 6 ft 7½ in. high from top of rail. Couplers are of railway type, one fixed and the other free to rotate to allow the cars to be dumped without uncoupling. Fig. 31 is a view of a 260-cu ft car being dumped; the tippie operation is automatic with push button control. The 188-cu ft Granby-type cars are 13 ft long center to center of coupler, 7 ft wide overall, and 6 ft 8 in. above the top of rail.

Fig. 32 is a view of a standard 16-car ore train at Frood-Stobie. Three of these trains operate on a main level.

Because of the difficulty of handling this equipment through the shafts, repairs to cars and locomotives are made in underground repair stations, see Fig. 33.

Power for trolley haulage is supplied by 300-kw Hewettic rectifiers, see Fig. 34, which convert power from 2200 v ac to 250 v dc.

Track is standard 85-lb rail at 36-in. gage, laid on 7-in. flatted pine ties and graded at 0.4 pct in favor of the ore load. The track is offset 9 in. from center of drift; walkway clearance is 2 ft 9 in. and clearance on the opposite side is 9 in. Turnouts from the main haulage ways are of 120-ft radius; the minimum radius of other turnouts is 58 ft. The trolley

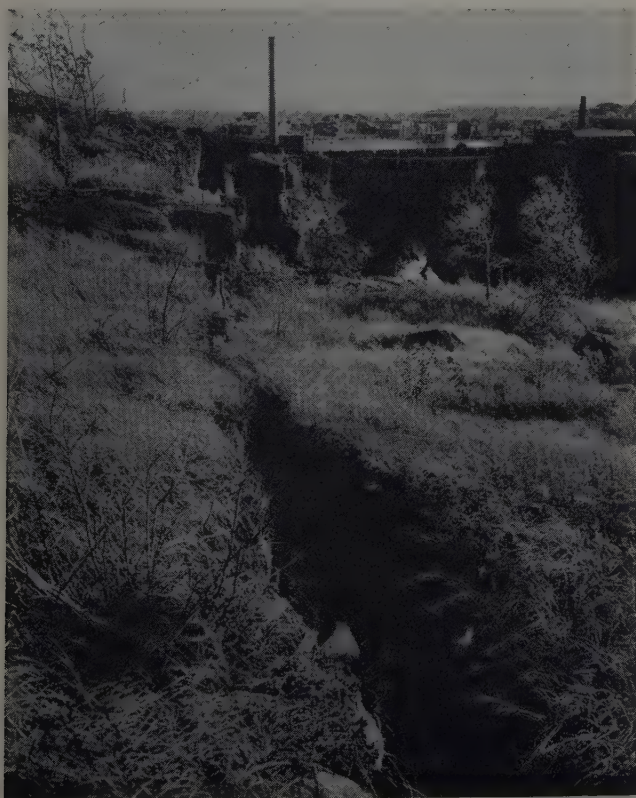


Fig. 39—Crack at surface where slice of ore is breaking away from solid.



Fig. 40—Ore disintegration through caving action.

line, 3 ft from center of track, has a 4/0 trolley wire and 4/0 feeder on universal hangers fastened to the drift back support beams. Trolley guards are 1x6-in. boards clamped to the hangers.

Loading Chutes: Chutes, constructed of concrete and steel, are 5 ft 10 in. wide, with a 4-ft brow opening, see Fig. 35, and lined with 2½-in. carbon steel billets. The ore is controlled by air-operated undercut, arc gate, and chain curtain. The curtain is constructed of six lengths of 2-in. anchor chain weighted with 1½x2-ft steel billets. The gates are cast steel and the operating cylinders are 8-in. diam with 11-in. stroke.

Ventilation: Similar systems of ventilation are used in the Frood and Stobie sections of the Frood-Stobie mine and at Murray.

Efficient ventilation is provided at each mine by a balanced system which has direct connections to collecting drifts at the tail pulley end of each slusher drift so that fresh air enters through the slusher station.

The systems were designed to supply 10,000 cfm per operating slusher drift, with an additional 50 pct for prevention of leakage through non-operating drifts and for the ventilation of development headings.

At the Stobie section of the Frood-Stobie mine where 750,000 cfm will eventually be required, there will be a main intake and three main returns with an axial flow fan mounted at the collar of each raise.

The main intake is a raw raise 21 ft in diam. Two of the returns are also circular raw raises, one being 15 ft in diam to handle 300,000 cfm, the other 17 ft in diam to handle 400,000 cfm; the third return is a raw elliptical raise 9½x16 ft to handle 200,000 cfm. Main airways in the slushing horizons are 10x12 ft, and connections to the main returns are 6x7-ft, 8x9-ft, and 9x10-ft drifts, depending on the volume to be handled.

Normally 75 pct of the fresh air for the stoping area on a level is supplied directly from the intake airway to the slusher drifts through controls at the drift intakes, and the balance is supplied through service raises to the sublevel drill drifts. From these drifts the air downcasts through the stopes to the slusher drifts where it joins the main flow. Controls in the slusher drift exhaust connection are closed when the slusher drift is not operating.

For primary blasting a strong downcast through

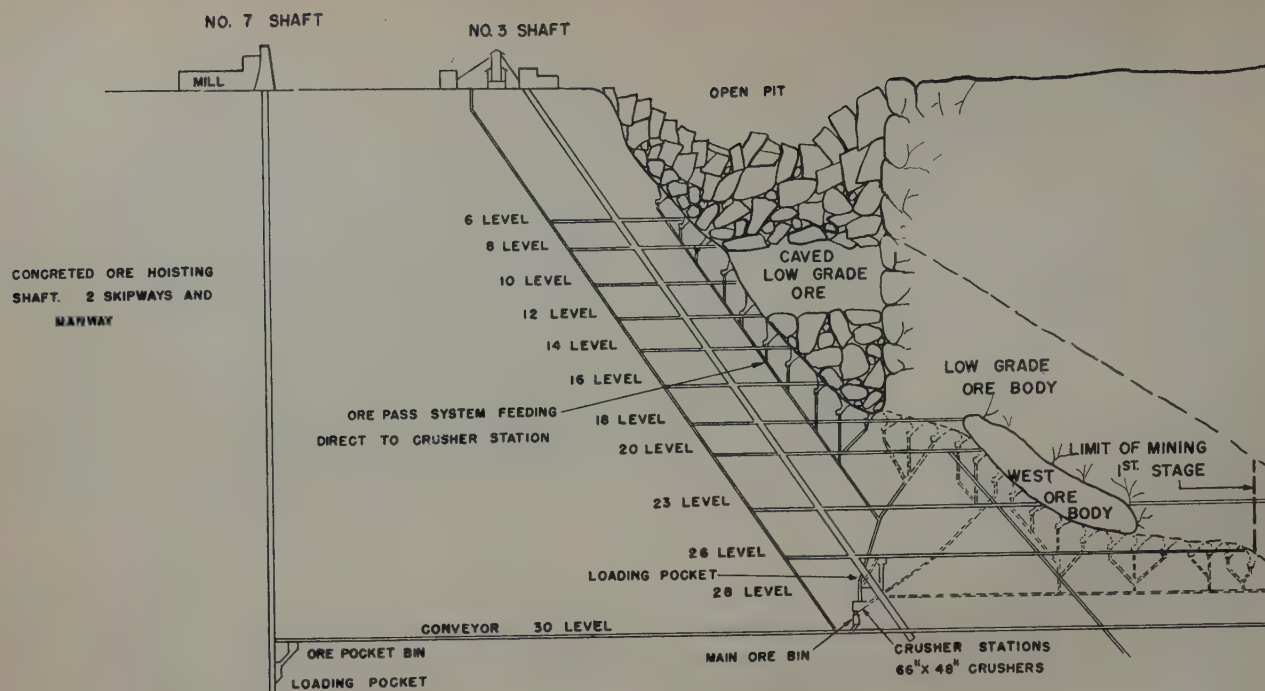


Fig. 41—Diagrammatic section, first stage of caving mining.

the stopes is provided by reducing the volume going direct to the slusher drifts and supplying more air through the drill drifts.

Mining by Induced Caving

The caving method of mining is being used on the upper levels at Creighton mine for the recovery of ore that had long been considered too low in grade to be profitably mined. However, studies and research followed by production on a small scale showed that by using the induced caving method of mining, and concentrating the ore at the mine to reduce transportation costs, recovery of the low grade ore was possible. Consequently when the need arose for quick expansion of underground ore production, planning and development of the large-scale caving project, now producing 10,000 tons per day, was immediately undertaken.

A new ore-hoisting shaft was needed to handle the production from the caving operation. No. 7 shaft, designed for ore hoisting only, was sunk initially to a depth of 2056 ft. It is concrete-lined throughout and has two skip compartments separated by a manway and pipe compartment, an arrangement that allows the hoist to be set up at a minimum distance from the shaft.

Serving No. 7 shaft is a 14-ft 110-in. parallel double-drum geared hoist driven by two 2750-hp, 600-v, 500-rpm dc motors with rotating-type control. The skips have a capacity of 15 tons each and the hoisting ropes are 2-in. flattened-strand with a breaking strain of 380,000 lb. Hoist operation is automatic with push button control at the loading pocket. Hoisting capacity with Kimberley skips averages 650 tons per hr.

The No. 7 shaft headframe, see Figs. 36 and 37, is a concrete structure 197 ft high, and both the headframe and the hoisthouses are integral parts of the concentrator building; ore from the receiving bin in the headframe is fed directly to the concentrator crushing plant. From the concentrator, the

bulk concentrate is being pumped through a 7½-mile pipe line, see Fig. 38, to the reduction plant at Copper Cliff.

Underground the initial program included the construction of a crusher station, two concrete and steel-lined bins for crushed ore storage, and an 1850-ft conveyor, as well as the driving of ore pass systems, slusher drifts, boxholes, service raises, and ventilation drifts, and raises needed before the first production could be obtained from the caving area. Further development includes the construction of a second crusher station with ore bin and conveyor, and the extension of the production workings along the strike and down the dip of the orebody.

The ore being mined by caving is the disseminated norite that makes up the major portion of the Creighton ore zone. The dissemination becomes more sparse in the hanging-wall direction and mining will be done to a cut-off grade. The ore occurs as a sheet, with mineable widths up to 600 ft, overlying and forming the hanging wall of the breccia and stringer orebodies. The average dip is 45° and the length along strike is 2200 ft.

An important factor in the successful application of the caving method to the Creighton low grade ore is the occurrence of many intersecting shears and joints. Studies have shown that the average frequency of jointing along any given line is one per ft and that 90 pct of the joints have plane surfaces on which the bonding is weak. Once tensional stresses are set up through undercutting and ore drawing the ore disintegrates, breaking along the joint planes into pieces small enough to be readily handled.

Early Mining: Mining was started at Creighton in 1901 and has conformed closely to the general pattern followed throughout the district. The first ore was produced from an open pit in the outcrop of the breccia ore. When it became necessary to go underground the ore was mined first by open stoping methods, heading, and bench for two levels

below the pit, followed as the mine grew deeper by shrinkage stoping, then by cut-and-fill and square-set stoping.

The open pit reached a final depth of 200 ft. Shrinkage stoping was carried on down to the 40th level, 2600 ft below surface. Below the 40th level all mining has been done by square-set stoping.

From surface to 30 level the breccia ores have been completely mined out. The disseminated ore overlying the mined out area from 28 level to surface will be recovered in the first stage of caving mining.

In this area no floor pillars were left. The vertical pillars were broken up while still surrounded with shrinkage ore. As the broken ore was drawn from the stopes the hanging wall caved gradually and the stopes became filled with caved, disseminated norite.

Pattern of Subsidence: Of particular significance to the present caving operation is the relationship between hanging wall subsidence and mining in the shrinkage area.

In this area there was rapid withdrawal of the ore, especially in the upper levels, so that lateral support was withdrawn from the hanging wall. Slices 100 to 200 ft wide and 400 to 800 ft long were cut off by cracks that developed and widened laterally with no appreciable vertical movement, see Fig. 39, indicating that failure was due to tensional stresses rather than shearing. Once the slice was broken clear, subsidence began and far outstripped the horizontal movement.

At first the face of a slice was fresh and solid, but within a few months interlacing cracks developed within the slice, and as drawing continued tension and torsion combined to open slips and joints till the mass lost cohesion and disintegrated, Fig. 40.

Surveys showed that the thickness of the slices that broke away corresponded to the width of the stoping area from which the ore was being with-

drawn. The hanging wall limit of a slice was always vertically above the hanging wall of these stopes. Subsidence at surface has never extended beyond the vertical above the hanging wall limit or the end limits of the area mined out by open stopes.

To induce caving beyond the present limits, the low grade ore will be undercut along the strike and down the dip of the orebody. Undercutting will be done by the blasthole method. Starting at the face of the caved area, panels will be undercut along the lines of boxholes from the slusher drifts. The rib pillars between the panels will be blasted out in sequence from the bottom of a section towards the top along the dip of the orebody.

Fringe stopes are not considered necessary. Experience indicates that as the broken ore in the caved area is drawn, the ore above a new undercut will be put in tension and will crack through to surface breaking away from the solid and disintegrating as drawing is continued.

Experimental Work: The manner in which the hanging wall subsided and the rock disintegrated indicated the possibilities of caving mining at Creighton, but many problems remained to be solved.

It was apparent that mining the low grade ore would be simplified in some ways and complicated in others because the higher grade ore underlying the low grade had been mined out. The low grade ore was undercut except where there was overlapping at the ends of the ore zone, so that many millions of tons of caved ore lay ready to be drawn. On the other hand the old mine openings would interfere with an orderly arrangement of openings for the new program. Some of the problems were solved in the laboratory by experiments in which scale models were used that embodied the factors anticipated in actual underground mining while others incapable of solution on a laboratory scale were worked out in pilot operations underground.



Fig. 42—Composite plan, first stage of caving mining.

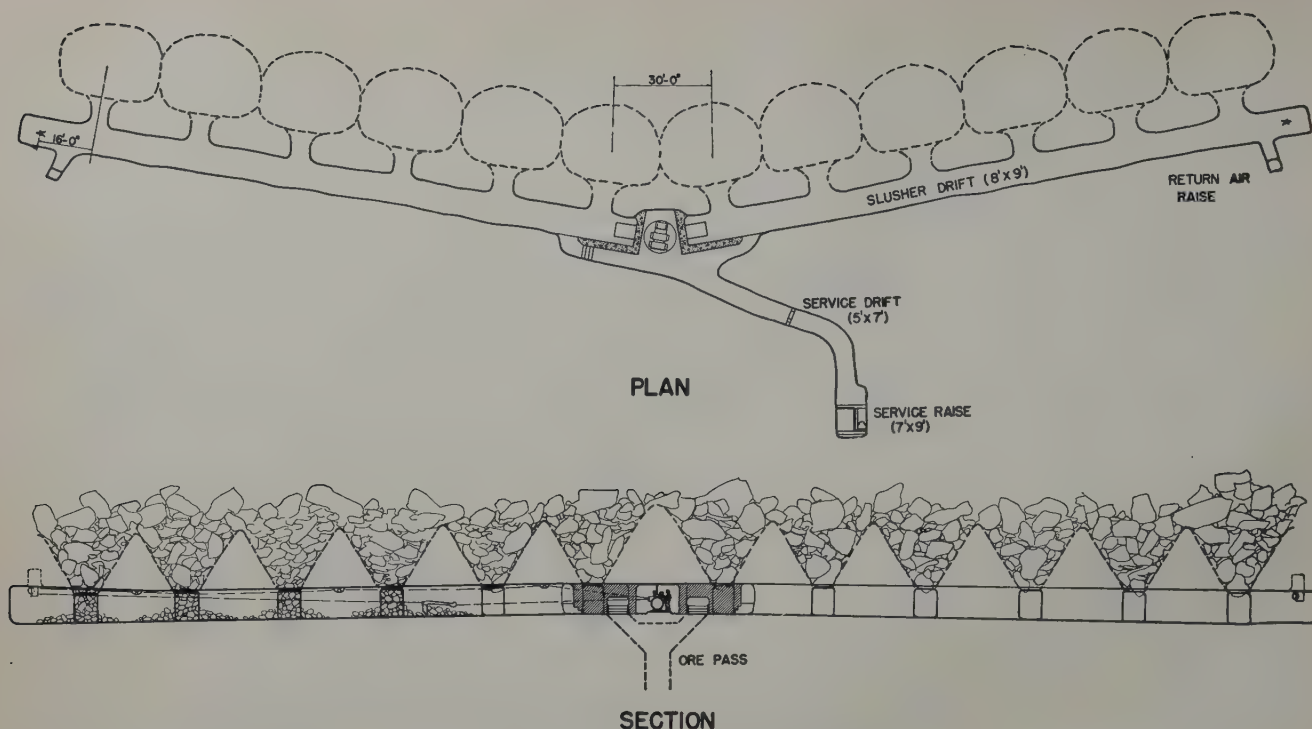


Fig. 43—General arrangement for two-way slushing.

It was decided that any of the old openings which interfered with the new development program could be walled off by concrete bulkheads.

Experimental work on ore drawing in caving showed that as ore is drawn from a stope chute the draw pattern in the stope for a height of 200 ft vertically above the chute is cone-shaped, the slope angle of the sides of the cone being 77° . Above 200 ft the pattern is cylindrical, the cylinder having a 125-ft diam.

Experiments also demonstrated that to minimize dilution it is better to draw small quantities of ore from each chute in rotation. This lowers the mass of broken ore without disturbing the contact between the ore and the capping. Concentrated drawing of a few chutes causes early dilution. It was also shown that boundary chutes should be drawn more heavily than internal chutes when the boundary is ore in place, and less heavily than internal chutes when the boundary is caved capping.

General Layout: The low grade ore for a height of 1750 ft from 28 level to surface is being recovered in the first stage of the caving program. Fig. 41 is a diagrammatic section showing the first stage of caving mining.

The ore flows by gravity to the 28 level where the primary crushers are located. They are 66x48-in. crushers driven by 300-hp motors and have a capacity of 590 tons per hr crushing to 6 in. From each crusher the ore falls to a 1500-ton concrete-lined bin on 30 level and is fed from the bin to a 48-in. conveyor carrying the ore 1850 ft to the loading station storage bin at No. 7 shaft.

The orebody has been divided along the strike into six 350-ft mining sections, see Fig. 42. For each section there is a service raise system, a ventilation raise system, and a main ore pass with 20 or more branches leading to as many slusher drifts in the footwall.

The ore pass is a compound raise which, starting from the crusher station, is made up of 75 to 135 ft

of $37\frac{1}{2}^\circ$ slide, 250 to 400 ft of 65° raise, and 1000 to 1200 ft of 47° raise with branches to the slusher drifts. Each pass is on the center line of a mining section. The passes feed to the crusher stations and each of the long slides at the bottom reaches across from the main part of the pass to a station. The slides are concreted and are lined with 3-in. nickel alloy steel wear plates. The upper portion of the passes is made flat to keep the branch raises short, and because of the low inclination no ore is allowed to accumulate there. Storage is in the 65° portion and varies from 1000 to 2000 tons.

Slusher drifts, designed for two-way slushing, are made up of two branches running parallel to the footwall in opposite directions from the ore pass. Each branch is approximately 175 ft long. The drifts are spaced in plan at 40-ft centers and the vertical distance between them varies with the inclination of the footwall. The boxholes are at 30-ft centers along the hanging wall side of the drifts and the slusher stations are in the center directly over the branch ore pass.

The service raise system is made up of a series of raises each approximately 500 ft long. The raises are 50 ft from the slusher drifts on the footwall side and are connected by service drifts to the slusher stations on all slusher sublevels.

The collecting raises of the ventilation system are in the footwall on the boundaries between the mining sections and are connected to opposite branches of the slusher drifts in two mining sections.

The original mining between 2800 level and surface was done through No. 3 shaft, which has been reconditioned and is being used for handling men and supplies. The main levels from No. 3 shaft were driven at 120-ft intervals with sublevels between where necessary. Much of the old development from No. 3 shaft is being used in the new mining program.

The two central mining sections in the low grade area are now in production, and mining will be

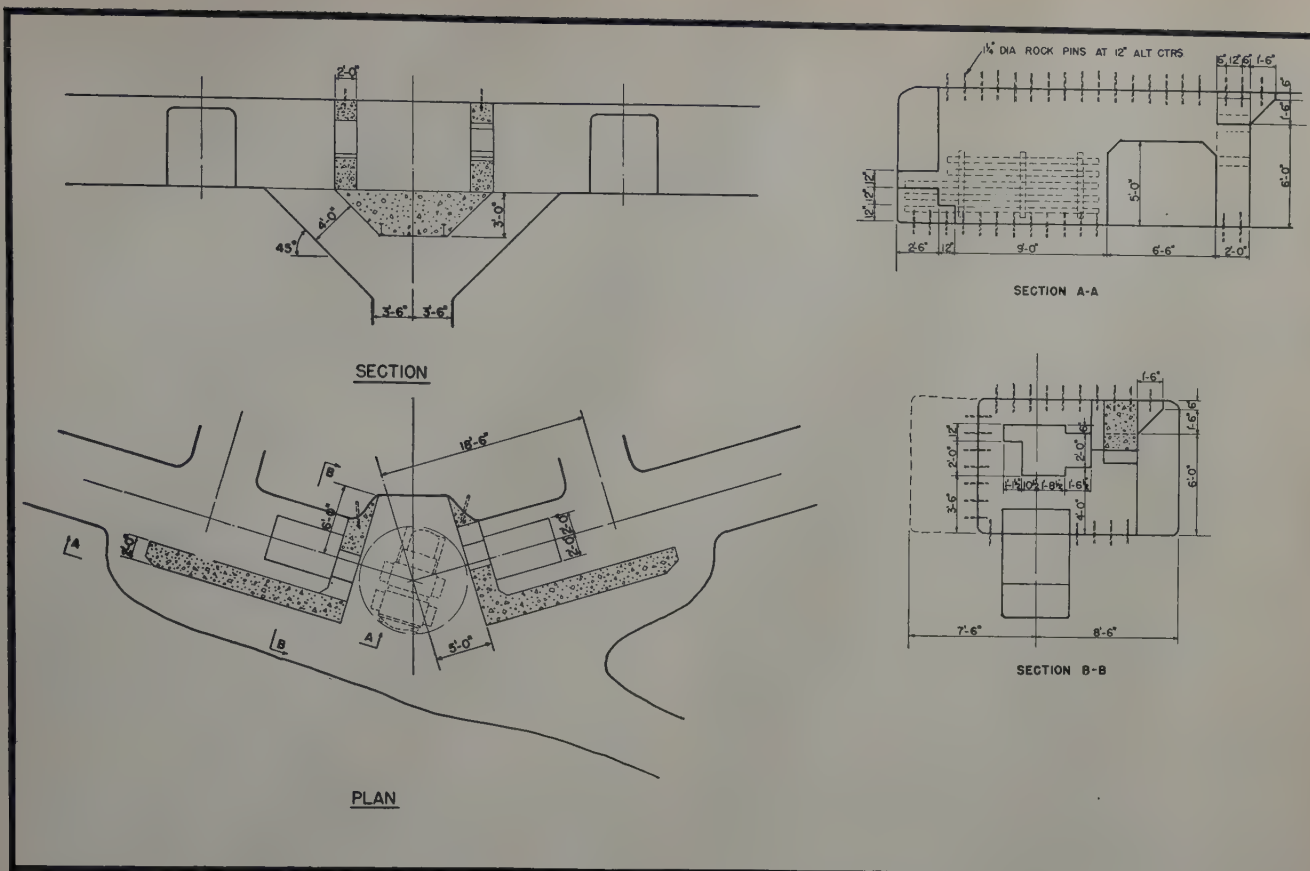


Fig. 44—Detail of slusher station and discharge openings.

started in the other sections as quickly as possible, since it is planned to draw the caved ore on a continuous front along the full length of the orebody to avoid the dilution that occurs where boundaries are formed between waste and ore.

Development: In each section development is completed over the entire area served by an ore pass so that when the section goes into production there will be a minimum dilution of the ore by development waste. The large amount of preparation required before any production is possible makes it necessary to concentrate this work on one mining section at a time. From 20 to 25 slusher drifts must be developed simultaneously. The ore pass and service raise are driven first. They are divided into lifts of 300 ft driven simultaneously from service drifts extended from existing footwall drifts on the old levels. Each lift is driven as a two-compartment stulted raise, containing a manway compartment and a chute compartment. The manway in the raise is above the chute and 5-in. flatted pine stulls slung in steel stirrups are placed horizontally at 5½-ft centers on a line 3 ft from the back of the raise to support the ladders and the timber slide. A main ore pass raise is 7 ft wide and 9 ft high, the branches being 7 ft square except for the last 10 ft, which are 4x4 ft square.

The service raises are 9 ft wide and 7 ft high. They are retimbered upon completion and divided into a manway compartment and a compartment for hoisting supplies. The permanent timber consists of 8-in. flatted pine footwall plates and dividers with 3-in. plank brattice nailed to the dividers between compartments. The track in the hoisting compartment is 40-lb rail at 24-in. gage.

The service drifts on the slusher sublevels are driven 5x7 ft. They are collared as the raise is advanced, and after the raise is completed they are connected to the branch raises at the slusher station locations. To protect the service raise from the effects of blasting a turn is made about midway along each drift; in addition a baffle is constructed consisting of 8x10 timbers placed vertically and dapped into 8-in. channels in concrete at the floor and back of the drift. The channels have a short section of flange cut away for easy changing of the timbers.

Slusher drifts 8x9 ft are driven and the boxholes completed before slusher station construction is started. Boxholes are normally 18 ft long and 8x12 ft at the throat and are belled where they enter the ore zone. A boxhole more than 50 ft long is driven as a 7x11-ft cribbed raise, and a 4x6-ft blasting drift with 4x4-ft manway raise from the slusher drift established on the footwall side of the boxhole.

Slusher Drifts and Equipment: The general arrangement of a station and slusher drifts is shown in Fig. 43. The detail of the slusher station and discharge openings is shown in Fig. 44.

The slusher discharge openings for each of the drifts are made by widening the 4x4-ft portion of the ore pass below the break through to form a V 4 ft wide and 26 ft long with ends sloping to the raise at 45°. The center of the V is bridged with a concrete slab 3 ft thick which becomes the floor of the slusher station and the foundation for the slusher hoist. Two L-shaped concrete walls enclose the discharge ends of the slusher drifts and separate them from the station. The slusher hoist

is mounted on a turntable and clearance of 10½ ft between the end walls allows it to be turned through 180°. The walls of the station are 2 ft thick and are anchored to the rock with steel pins. The side walls are lined for 3 ft from the bottom with salvage 60-lb rail placed horizontally at 9-in. centers. Openings for the slusher ropes are left in the end walls. Opposite the slusher discharge, openings left in the side walls for moving equipment in and out of the slusher drift and for convenience in barring are closed to a height of 3 ft above the floor with 5-in. flatted timbers held in slots in the concrete at the sides of the openings.

The entrance to each slusherway is at the end of the side wall over a 3-ft curb that prevents ore from spilling out into the travelway. A stairway on the outside and toeholds and handholds inside at the entrance are provided for climbing in and out of the drift.

The slusher hoist turntable arrangement utilizes castings. With a 1-ft skid plate bolted to the bottom of the frame the hoist is mounted on a set of radial castings anchored to the floor slab. The castings are machined to give a smooth surface under the skid plate and are carefully levelled and grouted in place. The hoist and skid plate are anchored to the castings by bolts inserted from the top through both the hoist frame and the plate. The nuts for the hoist anchor bolts are elongated to keep them from turning and to provide good bearing against the shoulders of the T-slots in the castings. To keep the hoist centered as it is swung around, the skid plate turns about a pivot pin at the center.

The electrical connection to the motor is made with a piece of 4000-v rubber-covered flexible power cable long enough to permit the hoist to be turned without disconnecting the motor leads.

Slushing equipment for caving is the same as that used in sublevel blasthole stoping.

The electrical equipment is located in control rooms on the main levels far enough away from slusher-drift blasting to be protected from concussion. The circuit breakers are motor-operated and as many as eight hoists may be controlled from one switch room. The power cables and control cables reach the hoists through the service raises and manway drifts or through boreholes.

All blasts are fired electrically. For blasting in several drifts simultaneously a permanent ungrounded 110-v blasting circuit is installed in each service raise with junction boxes at the slusher drifts and a standard shot-firing switch at each of the slusher hoist switch rooms. When a blast is confined to one drift a 10-shot blasting machine is used.

Operation of Slusherways: The crew consists of a slusherman and a helper for each slusherway. Pieces of ore too large to be slushed into the raise are broken in the drift by sand blasting and hang-ups in the boxholes are brought down by pole blasts.

For draw control the slusherman records the number of scraper loads of ore taken from each boxhole. Approximately 2000 tons are slushed from each of six boxholes in one branch of the drift before the hoist is turned, and an equal tonnage drawn from the boxholes in the opposite branch. The hoist must be turned about once a month. The slusherman and his helper, using the tugger hoist mounted on the slusher and a set of double blocks, can complete the operation in about 2 hr.

Slusher hoist greasing and inspection is done by special service men who are directly responsible to

the mine supervision. Major repairs are done by electrical and mechanical department personnel.

Ventilation: Efficient ventilation is provided by an exhaust system which has direct connections to the collecting raises at the tail-pulley end of each branch of the slusher drift so that fresh air enters through the slusher station.

The system was designed to exhaust 10,000 cfm per slusher drift operating. The fan is mounted on the collar of the return air raise. It is a 124-in. diam axial flow fan rated at 300,000 cfm against 5 in. of water at 440 rpm and is driven by a 350-hp totally enclosed pipe-ventilated vertical motor. The impeller is hung below the motor on the fan shaft extension and is equipped with stainless steel adjustable pitch blades which can be varied to operate over a range of 2¼ to 5½ in. of water at rated volume.

From the top slushing horizon to surface the main return air raise is circular and 15 ft in diam. It is connected on 5 level by 10x12-drifts to two return air raises 120 sq ft in cross section that will also handle the exhaust air from the second stage of the caving program. The other return raises, for use in only the first stage of the program, vary in size from 5x5 ft to 7x9 ft. The transfer drifts, joining these raises to the central returns, are 8x9 ft and 9x10 ft.

Fresh air from surface is drawn down through the caved area and footwall development to the various levels and reaches the slusher station through the connecting manways and supplyways on the slusher sublevels. The flow of air through each operating slusher drift averages 175 ft per min.

In the exhaust connection from each drift a reinforced concrete bulkhead is installed, with an opening which can be varied in size to maintain an average flow of 10,000 cfm. The opening is closed to travel by a steel rod grid and is provided with a cover of salvage conveyor belting that is closed during periods when the slusher drift is not in operation. When mining from a drift is completed the bulkhead is sealed with a steel plate.

The system of return airways is entirely separate from all operating development openings, and a high efficiency is maintained without difficulty.

Drainage: Surface water that runs into the caved area drains through the boxholes into the slusher drifts. If allowed to enter the ore passes it causes great difficulty in handling the ore.

The slusher drifts are graded at 1½ pct down from the discharge openings and the water flows through the ventilation raises to the mine drainage system. Any pools of water which collect between operating periods are pumped dry before slushing is resumed.

Sequence of Extraction: In each mining section 5 to 8 slusher drifts in sequence are in operation at one time. Commencing with the upper sublevels, each drift down the footwall is depleted in turn, to be replaced by a new drift brought into production at the bottom of the sequence.

The capping follows the descending ore, forming a plane extending from the hanging wall above the lowest producing slusherway to the footwall. This plane of retreat is kept as uniform and as flat as possible and its pitch is determined by the number of producing slusher drifts.

Scale models based on the recorded tonnages drawn from each boxhole are used to control the inclination and uniformity of the plane of retreat.

Sponge Iron at Anaconda

by Frederick F. Frick

SPONGE iron as produced at Anaconda is a fine, —35 mesh, impure product, about 50 pct metallic iron, obtained from the reduction of iron calcine at a temperature of 1850°F by use of coke resulting from slack coal. The metallic iron particles are bulky and spongy and precipitate copper readily and rapidly from a copper sulphate solution. Investigation of the treatment of Greater Butte Project, Kelley, ore at Anaconda early showed the desirability of using sponge iron as a precipitant for the copper in solution resulting from desliming of the ore in a dilute sulphuric acid solution.

Anaconda had done considerable work on the production of sponge iron in 1914 for use as a precipitant of copper from leach solutions. Some success and considerable experience were attained at the time, indicating that sponge iron might be successfully made by a modification of the process used in 1914, a batch process in which an iron calcine was reduced by means of soft coke, resulting from noncoking coal, in a Bruckner-type revolving horizontal cylindrical furnace widely used 50 years ago. The coke and calcine formed the bed in the Bruckner furnace, which was rotated at about 1 rpm. The bed was brought to a temperature of about 1800°F by means of an oil flame over the surface. Although results were reasonably satisfactory, they did not warrant full development of the process at that time.

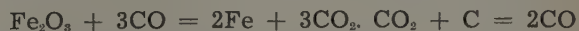
A good deal of work has been done in the last 50 years on the production of sponge iron. The objective in some cases has been the production of a precipitant for copper from solution, but the bulk of the work has been done for the production of open-hearth steel furnace stock. The production of an open-hearth stock presents two problems rather than one: first, production of the sponge iron, and second, what is perhaps of equal difficulty and importance, conversion of the sponge iron into a form suitable for use in the open-hearth furnace. So far as is known to the writer, none of the sponge iron processes tried in the past have proved to be economically feasible. However, Anaconda had a combination of conditions appearing to justify an attempt to produce sponge iron which would serve for the leach-precipitation-float process. It was thought that the process used in 1914, if changed to a continuous one, might work out satisfactorily. The following favorable conditions at Anaconda justified the investigation: 1—A sufficient tonnage of good grade iron calcine resulting from the roasting of a pyrite concentrate in one of

the acid plants, at substantially no cost. 2—Reasonably cheap natural gas. 3—The fact that there was no need for production of a high grade product. 4—The fact that there was no need for obtaining a consistently high reduction of the iron in calcine.

A small revolving Bruckner-type furnace about 2 ft ID by 4 ft long was set up for early pilot work at the research building. This pilot furnace showed that a satisfactory product could be obtained at reasonable cost. It also indicated a marked advantage in preceding the reduction furnace with a furnace of similar size and capacity for preheating and roasting out any residual sulphur from the feed. The small furnace was operated for several months, various details of the process were worked out, and sponge iron was produced to supply a pilot LPF plant which treated 300 lb of Kelley ore per hr. Later a second pilot furnace 5 ft in diam and 12 ft long inside was set up at our reverberatory furnace building. This furnace confirmed the data of the small furnace and gave a basis for design of the final plant.

At Anaconda a pyrite concentrate, running about 48 pct S, is recovered from copper concentrator tailings by flotation. This concentrate is roasted to sulphur of 3 pct or less at the Chamber acid plant. The iron calcine contains about 57 pct Fe and 18 pct insoluble.

The iron calcine feed, as mentioned before, is re-roasted and preheated in a re-roast furnace preceding the reduction furnace. Both are of the Bruckner type. The re-roasted calcine is fed into the reduction furnace at 800° to 1000°F along with 30 pct slack coal. In the feed end of the furnace the volatile is burned from the slack, giving a soft coke which readily serves for reduction of the iron. Hard metallurgical coke will not serve the purpose, since it does not reduce CO₂ readily at a temperature of 1850°F. All indications are that the actual reduction of the iron is accomplished by carbon monoxide below the surface of the bed, which is 30 in. deep at its center. Apparently there is a constant interchange:



Actually iron oxide is reduced by CO at somewhat lower temperature than the 1850°F used in the process, but this temperature is necessary to obtain a satisfactory rate of furnace production. The furnace atmosphere is generally reducing, and typical blue carbon monoxide flames satisfactorily cover the bed. Gas flames from four 3-in. Denver Fire Clay Inspirator burners are played directly on the bed, which is slowly cascaded by the 1 rpm of the furnace. An excess of coke is necessary to assure maintenance of good reducing conditions in the furnace bed. Part of this coke is recovered for re-use.

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Discussion on this paper, TP 3472B, may be sent (2 copies) to AIME before March 31, 1953. Manuscript, Oct. 2, 1952. Los Angeles Meeting, February 1953.

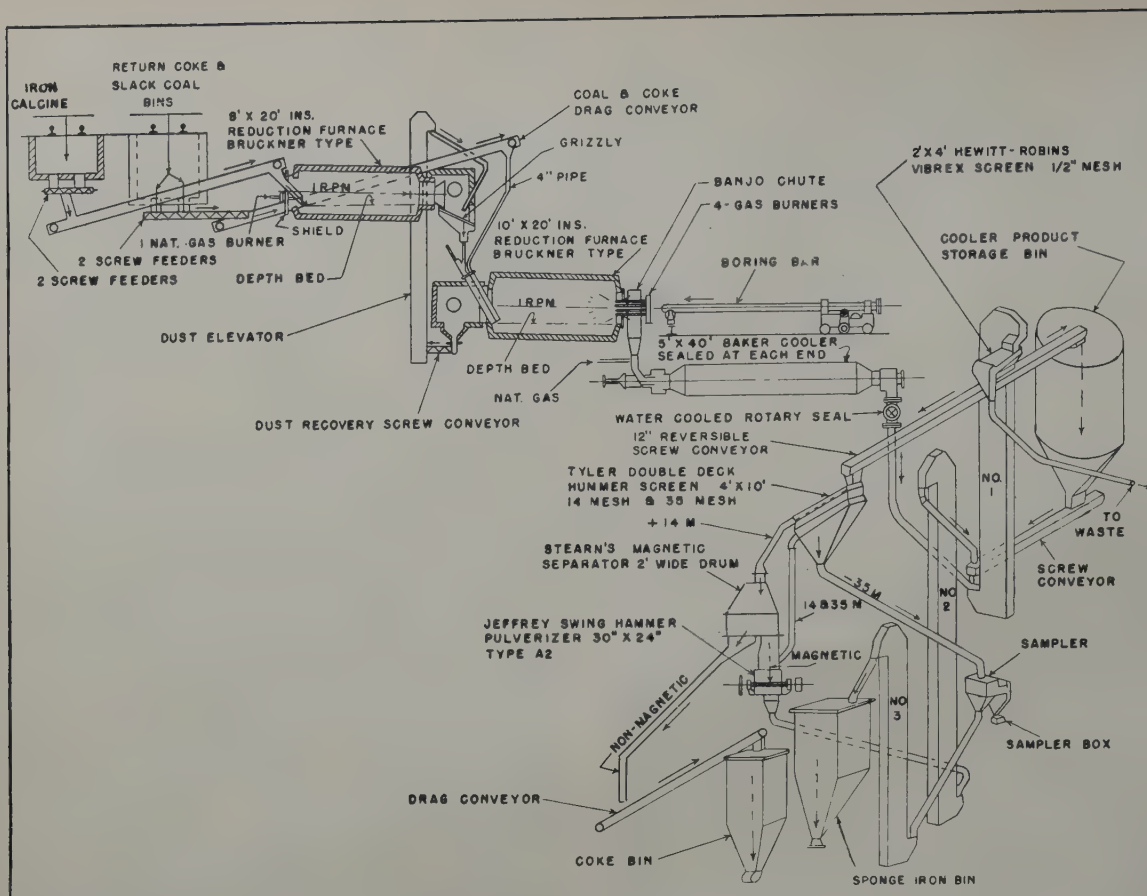


Fig. 1—Anaconda sponge iron plant.

A satisfactory final sponge iron product involves two steps, production of the iron, and prevention of reoxidation during cooling. The firing end of the furnace is almost entirely enclosed and the product overflows at the firing end into a receiving duct in which raw gas is passed to maintain a reducing atmosphere. The furnace product passes from the duct directly into a Baker cooler where it is cooled from about 1500°F down to about 150°F. The Baker cooler has rotary seals at both ends to exclude air and prevent reoxidation of the sponge iron. It has given marked satisfaction.

Table I. Production of Sponge Iron at Anaconda Copper Co.

Operating Data				
Tons calcine feed per day				65
Tons final product per day				45
Tons slack coal per day				20
Tons return coke per day				5
Mcf gas per day				300
Pct iron reduced				72
Bed temperature, degrees F				1850
Analyses				
	Fe, Pct	Fe ^o , Pct	S, Pct	Insoluble, Pct
Feed	58		0.50*	18
Final product	70	50	0.8	24

* Reroast to reduction furnace.

One of the essentials for satisfactory production of sponge iron is a low sulphur content of the feed to the reduction furnace. The sulphur content of the feed should not exceed 1 pct, see Table I. A larger amount gives wall accretions, ball formation in the bed, and a dark inferior product which gives a poor efficiency in precipitation of copper.

Both pilot plants indicated the need for a boring bar in a commercial plant. At the 1850°F temperature necessary for satisfactory production from an individual furnace, there is a tendency for the formation of accretions on the walls of the furnace, and it is not feasible to remove these accretions by hand barring. A mechanically operated water-cooled boring bar was designed which enables removal of the accretions in a very satisfactory manner. At the outer end of the boring bar there is a knife 1 ft wide which is brought to within about 3/4 in. of the brick lining of the reduction furnace, at the firing end. The knife is advanced horizontally at a rate of 1 ft per min. The revolution of the furnace gives the knife a spiral path on the inner surface of the furnace and the accretions are cleanly shaved off.

The cooled furnace product requires preparation for use, principally reduction in size to -35 mesh and recovery of excess coke for return to the reduction process. Size reduction is desirable to reduce the tendency of sponge iron to sand out in the flotation cells of the LPF plant. Processing, shown in Fig. 1, has presented no serious problems.

To summarize, the production of sponge iron has proved satisfactory both technically and economically. The capacity of the plant has not yet been determined. There is little doubt that the operation will be improved as time goes on.

Acknowledgment

The production of sponge iron was developed by the Research Department. The successful outcome of the project is due to the untiring efforts of F. L. Holderreed, Assistant Research Engineer.

Principles of Present-Day Dust Collectors And their Application To Mining and Metallurgical Industries

by J. M. Kane and R. H. Walpole

IN all probability the mining and metallurgical industry as a whole can demonstrate a larger economic return from installation of dust-control equipment than any other major industrial group. This fact has partially accounted for the marked increase of dust-control installations made during the past decade. While the primary objectives for installation of dust-collecting systems are improved working and operating conditions for men and equipment, the fact that an economic return can be anticipated on salvageable materials is an added advantage which shows in partial or complete equipment write-off. The conditions apply to most phases of the mining, milling, and smelting industry, both non-metallic and metallic.

As with any mechanical devices, selection of suitable dust collector equipment involves evaluation of available products with characteristics most nearly meeting conditions of the application at hand. When there is valuable product to be collected, and/or when there are possibilities of air pollution or public nuisance, collector selection is often guided by the maxim of "highest available collection efficiency at reasonable cost and reasonable maintenance." A brief review of dust collector designs will permit outlining of major characteristics of each group. Final selection will involve detailed data against a background of the problem under consideration.

The dry centrifugal collectors, see Fig. 1, represent a group of low cost units with minimum maintenance. They are subject to abrasion under heavy abrasive dust loads and to plugging with moist materials. Efficiency drops off rapidly on particle sizes below the 10 to 20 micron group. Because of the large amounts of -10 micron particles in most mining dust problems, they will normally be used as primary collectors and will be followed by high efficiency units. This combination is especially popular where the bulk of material is desired in a dry state with wet collection indicated for the final cleanup portion. In remote plant locations, dry centrifugals can be used alone if product in dust form has no value or if dust loading is light enough to eliminate a nuisance in the plant area. Where high efficiency dust collection equipment must be selected,

choice will normally involve fabric arresters, wet collectors, or high voltage Electro-Static precipitators.

Fabric arresters, see Fig. 2, rely on the passing of dust-laden air at low velocity through filter fabric. Velocity ranges from 1 to 3 fpm for the usual installation and may be as high as 10 to 20 fpm in arrangements where automatic frequent vibration or continuous cleaning of the filter media is employed. Fabric is normally suspended in either stocking type or in an envelope shape. Collection efficiency is excellent even on sub-micron particle sizes. Equipment is bulky, must be vibrated to remove the collected dust load, and is restricted in applications from temperature and moisture standpoints. Condensation of moisture on the fabric filter media causes plugging of the passages with great reduction in air flow. Temperatures for the usual medias of cotton or wool are 180° and 200°F maximum, although the introduction of synthetic materials such as nylon, orlon, and glass cloth have increased the possibilities of this type of collector for higher temperature applications.

The wet-type collector may employ a number of different principles so that entering dust particles in the gas stream are wetted and removed. Principles usually include impingement on collector surface or water droplets, often in combination with centrifugal forces. Variety of wet collector designs is indicated by typical collectors illustrated in Figs. 3 and 4. Collection efficiency is a function of the particular design, although the better collectors will have high collection efficiency on particles in the 1-micron range. Wet collectors have the advantage of handling hot or moist gases, take up small space, and eliminate secondary dust problems during the disposal of the material. At times collection of the material wet is a disadvantage. Wet collectors may also be subject to corrosion and freezing factors.

The high voltage Electro-Static precipitator, see Fig. 5, is probably the most expensive type of high efficiency collector. It finds its applications generally in problems in which collectors previously discussed cannot be employed. Its collection efficiency is based on its design features and can be excellent on the finest of fume particles. Material is normally collected dry. Gas temperatures are of no great concern as long as condensation does not occur within the dry type of precipitator and the temperatures do not exceed the limits for materials used in its construction. As with the fabric arrester, provisions

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Discussion on this paper, TP 3427B, may be sent (2 copies) to AIME before March 31, 1953. Manuscript, Feb. 20, 1952. New York Meeting, February 1952.

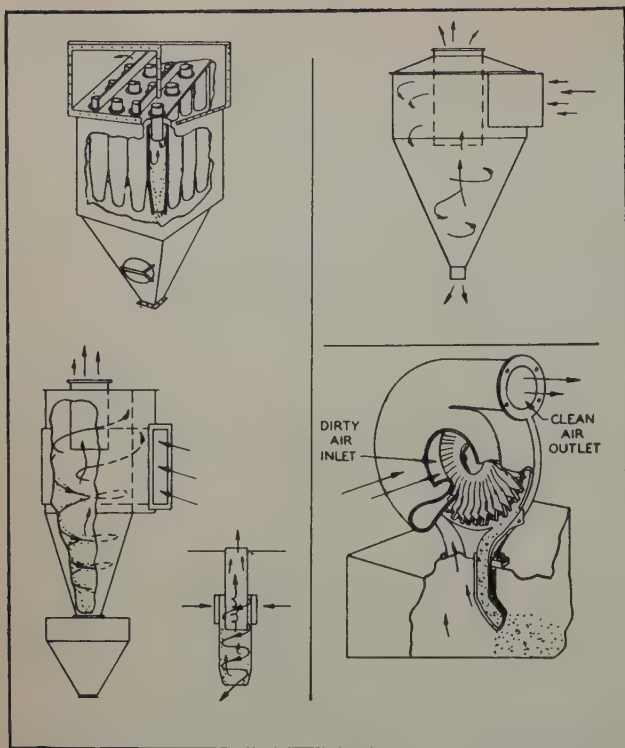


Fig. 1—Dry-type centrifugal collectors. Upper right, low pressure cyclone. Lower right, dry-type dynamic precipitator.

must be made for periodically removing the collected dust load and for disposing or transferring a dry product from the storage hoppers.

Handling of materials from the mine process through operations of secondary crushing, smelting, sintering, and roasting requires local exhaust ventilation to remove air-borne dust. Clean air reduces occupational hazards and permits ideal working conditions for maximum labor output. Dust is released during dry material transfer from one element of the system to a second and also from operations involving such processes as crushing and screening. The logical means of reducing atmospheric contamination is by localized control. This is achieved by enclosing the dust dispersion source and exhausting sufficient air from this hood or enclosure. Pring in a discussion of this subject effectively illustrated that there is a need for large enclosures and the proper location of exhaust connections, as can be noted from Fig. 6. With restricted hooding near the belt, excessive material will be picked by the exhaust system. The cushioning effect of breaking the fall of ore before it hits the belt can be noted by the need for additional exhaust at the back side of the chute in the latter case.

The degree of enclosure is frequently a function of the operation, generally a compromise between views of operating personnel who would like no enclosures to impede maintenance and inspection and those of the dust-control engineer who would like to house all equipment completely. Obviously complete enclosures would be the most desirable from a standpoint of dust-collecting requirements, but these are seldom possible because of required access, inspection, and material flow.

Consideration must be given to the cause of dust dispersion. For example, ore falling from the head of a conveyor to a feed point on a conveyor below sets up an air flow in the direction of fall. To prevent release of this volume with its load of dust

particles, exhaust volumes of from two to ten times the material volume must be expended in addition to the exhaust volume required to keep an indraft through all openings in the enclosure at the skirt boards of the second conveyor. The variables controlling the induced air volume are many and include such factors as size and surface area of the material, height of fall, and free-opening area surrounding the enclosure. The problem has been the subject of considerable field research, with indications that more accurate data for the guidance of the mining dust control engineer will be available in the near future.

A survey of many dust control applications covering material-handling operations in the mining group indicates the concentration of accompanying dust can vary widely. Factors such as system design, moisture content of mine run ore, and pickup velocities influence amounts of material to be handled by the dust collector. This may vary from 0.5 to 15.0 grains per cubic foot. Tests on typical dust collecting exhaust system controlling a secondary copper ore milling operation indicate concentrations of from 4 to 7 grains per cubic foot. Tests on a lead-zinc cone crusher exhaust system reported loadings of 15.0 grains per cubic foot.

By way of illustration, dust control benefits in the potash industry may be cited. Without an exhaust system, the dust escaping from conveyor belt transfer points handling finished potash completely filled a surrounding tunnel area. This necessitated daily removal of accumulated potash, requiring approximately 3 man-hr per day. Through control of air-borne dust at the source by local exhaust ventilation, the tunnel is now consistently clear, the collected material is redistributed on the conveyor belt, eliminating the possible contamination of the products, and 15 man-hr per week direct labor cost are saved.

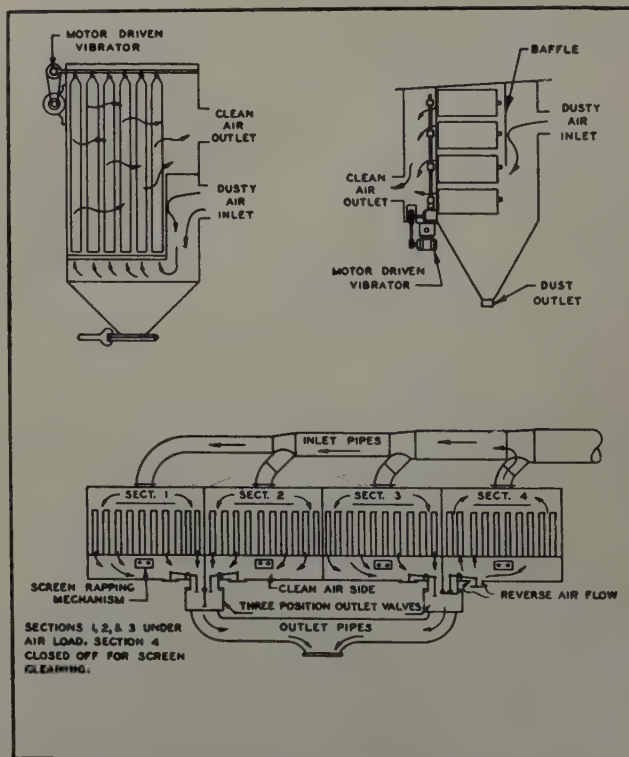


Fig. 2—Fabric arresters. The multiple section continuous automatic, the cloth tube or stocking type, lower left, and the envelope or bag type, upper right.

Some conveyor belt transfer points on smelting and roasting operations indicate concentrations down as low as 1 grain per cubic foot and as high as 10. The velocity at the enclosure takeoff frequently has a decided bearing on the dust concentration to be handled by the exhaust system. While the normal carrying velocity in an exhaust duct system is 3500 to 3700 fpm, if this velocity were maintained throughout the enclosure, the pickup rate would be excessive. Experience indicates that the takeoff velocity from hoods should be reduced to a point where only the airborne dust is removed. This is generally in the neighborhood of 500 to 700 fpm with an exhaust transition section designed to permit a gradual increase in velocity from this rate to the carrying velocity in the exhaust system ducts.

The selection of the type of collecting unit requires an analysis of the basic collecting units discussed above with specific reference to the problem being studied. In many applications in the mining and milling industry, material must be collected wet or dry, depending on the salvage method. This frequently is the governing factor for the selection of a wet or dry collecting unit. In screening and crushing operations, more emphasis is placed today on the transfer of collected material to central treatment areas by water, making the wet collector an effective unit for the sluicing of collected dust to settling pond, wet flotation plant, or even rock feed or storage. Where material can be collected dry and returned to the process by conveyor without upsetting production flow, the dry centrifugal or fabric-type collector finds application. While the electric precipitator type of unit offers the highest collection efficiency, its initial and operating cost is seldom justified by the particle size encountered in material handling systems. In most systems in this group, dry centrifugal and dynamic precipitating types of

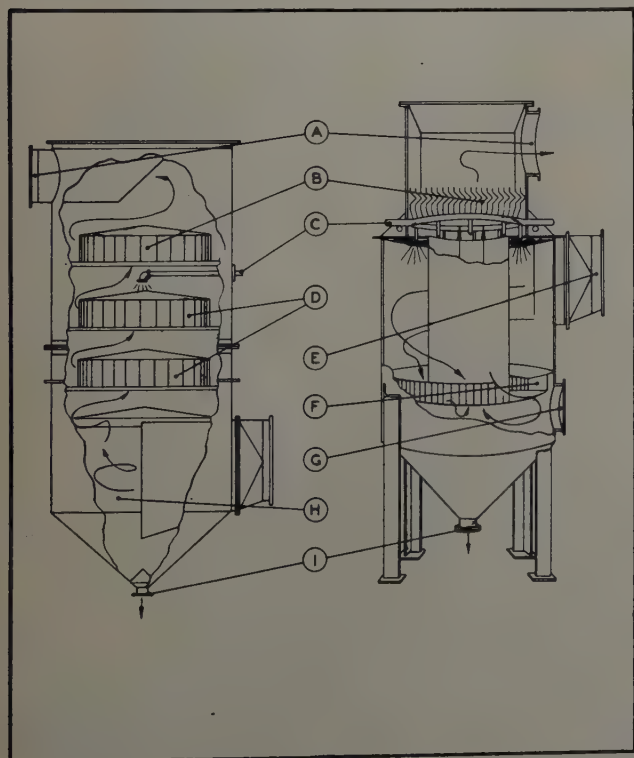


Fig. 3—Wet-type centrifugal collectors. A. Clean air outlet. B. Entrainment separator. C. Water inlet. D. Impingement plates. E. Dirty air inlet. F. Disintegrator. G. Inspection door. H. Wet cyclone for collecting heavy material. I. Water and sludge drain.

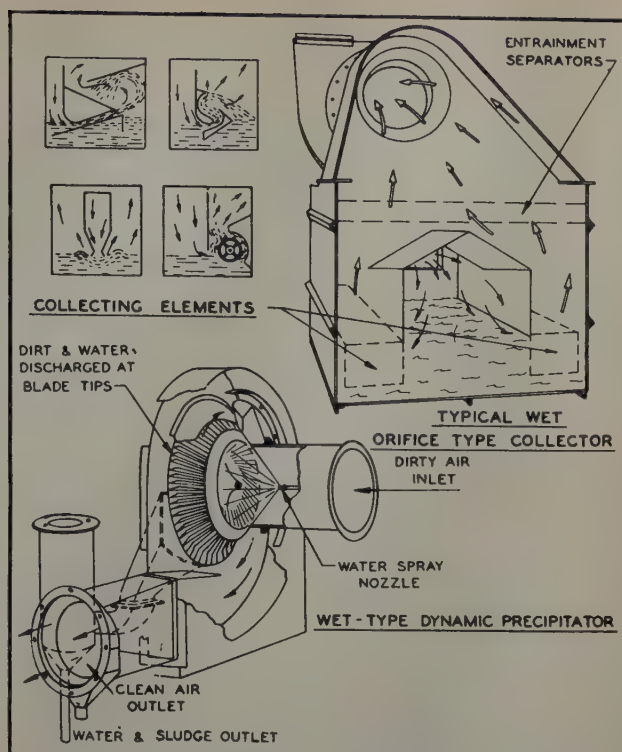


Fig. 4—Wet-type dust collectors.

units provide collection efficiencies in the order of 60 to 85 pct. The fabric-type collector and many wet-type collectors yield collection efficiencies of 98 pct and higher.

High Temperature Gas Cleaning

Dust control measures for driers, kilns, roasters, and furnaces introduce additional problems not encountered in material-handling dust control. In rotary units like the concentrate drier, kiln, or cooler, adequate air volumes must be handled within the drum to maintain proper temperatures. With restrictions imposed by the physical characteristics of the drum itself and the cascading of the material through the combustion gas stream, high pickup conditions usually exist. High temperature gases with their high moisture content introduce problems in the selection of suitable dust control equipment. While the dry centrifugal collector is the simplest approach to the collection of the escaped material, the lower efficiencies obtained usually restrict such collectors to service as a primary or scalping collector. Wet collection equipment is practical for many operations in this group. It is difficult to apply fabric collectors because of the moisture or temperatures encountered. Condensed moisture from the exhaust gas will cause plugging by filling the pores of the fabric itself, creating excessive pressure drop and reduced air volume.

With the metallic fumes released from roasters and sintering furnaces, and with the need for dry salvage of the material in the stack gases, the fabric collector or the electric precipitator is indicated and is used extensively in the lead and zinc field. Cooling of gases is necessary in either case with temperature limits on fabric arresters in the 180° to 200°F range. New media fabrics available on today's market, nylon, orlon, glass cloth, and others, may increase the potential use of this type of application. In many of these applications, the functioning of the wet-type collector is hampered by high corrosion rates with the release of sulphur compounds in many

of the roasting or furnace processes. There are, however, applications for wet-type collectors in the furnace and roaster exhaust field. Test data from copper anode electric furnace systems exhausted by local hoods and wet hydrostatic precipitators illustrate application of this type of equipment. Approximately 34 tons of copper oxide are collected each month of 8-hr operating days from the exhaust gases of two such furnaces. Exhaust volume is 21,000 cu ft at standard conditions. With 40 pct of this material less than 10 microns in size, collection efficiency of 95 pct is reported and makes possible an appreciable saving of copper oxide in addition to controlling a troublesome fume problem. Wet collectors, rubber-covered, have proved effective for roasting operations in uranium ores.

Lead Sintering

In the lead smelting field, the basic operation of the D and L sintering machine requires movement of air to sustain the process. The loss of salvageable material and atmospheric contamination require maximum dust control measures. The low efficiency of dry centrifugal and corrosion problems on wet-type collectors rule out these types. The fabric bag-house and electric precipitator are universally applied to this operation. Immediately following the sinter-machine process, crushing, quenching, and handling of sinter material take place. High dust loads varying from 3 to 10 grains are encountered because of the release of hot gases in handling of the porous clinker. Local exhaust control by hood enclosure over dust-releasing points and by wet-type collector systems has produced excellent results. Since the gases in the exhaust system are frequently below the dew point, duct accumulations often occur, regardless of conveying velocities. The sticky nature of the dust and the steam will rule out the application of dry collecting units. Test analyses indicate that collection efficiency better than 99 pct, reported as lead, can be achieved with good wet collectors.

At one plant, prior to the installation of an exhaust system collecting sintered dust from the escape gas,

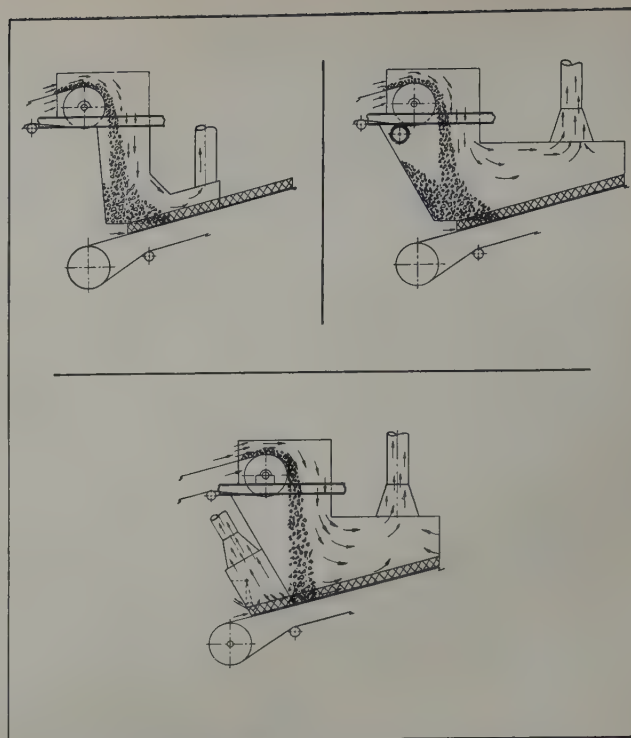


Fig. 6—Exhaust hooding for conveyor transfer points. In the poor design, upper left, and the improved design, upper right, the ore pocket breaks the fall on the belt. In the lower design the material falls directly on the belt. Two exhaust connections are required along the conveyor, none at the top.

corrosion was taking an annual toll of equipment and plant structures. The annual expenditure for building and equipment replacement exceeded the initial cost of the dust-collecting system itself. Furthermore, the material salvaged was value received and conservation of natural resources.

Dust Disposal and Salvage

The means of material salvage are varied within the industry. In the milling and sampling operations where flotation processes are accessible, the use of water to sluice collected material has proved successful. Since flotation operations generally have large water volumes available, slurry piping from collectors of the wetted dust is feasible. Open sluices have an advantage where gravity flow is used, as they are easily inspected for possible accumulations. The flow is generally to thickeners, then to flotation mills. Very frequently the value content in the dust is twice that of the mine-run ore. Where flotation is not accessible, the wetted material can be returned either directly to thickeners or through them to ore supply dumps without ore volume's appreciably increasing the moisture content of the raw ore. Another approach is to remove material from the collector in a sludge state, disposing it on conveyor belts ahead of secondary crushers or storage hoppers. Again, addition of this degree of moisture is not a factor of concern because of the low ration of water to raw ore supply. Where outside space permits, air drying of the sludge can be achieved, permitting the dried material to be reintroduced into the process by mixing with other supply materials. Slurry-type filters and filter presses are also used to remove moisture from sludge prior to rehandling of collected material.

In milling fields where dry collectors are employed, the collected material is returned from the process by conveyors or transfer chute.

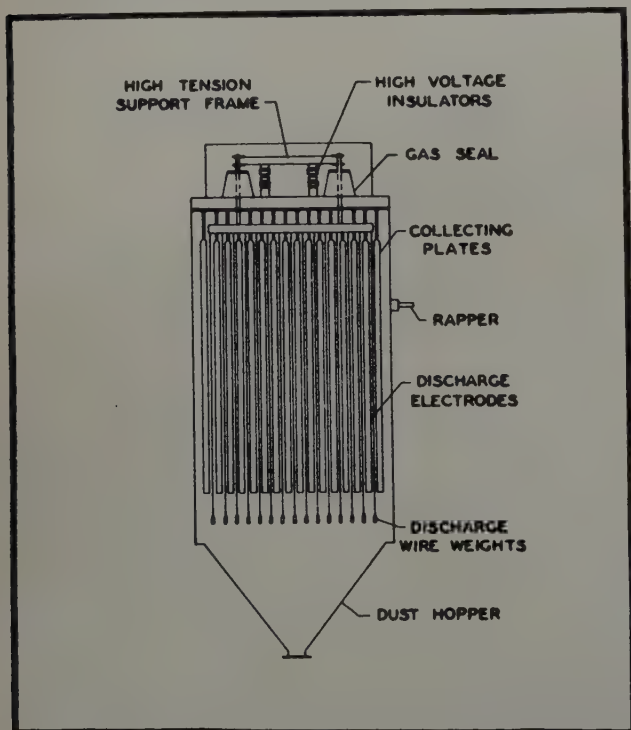


Fig. 5—High voltage electro-precipitator.

The United States Gypsum Company Mine, Heath, Montana

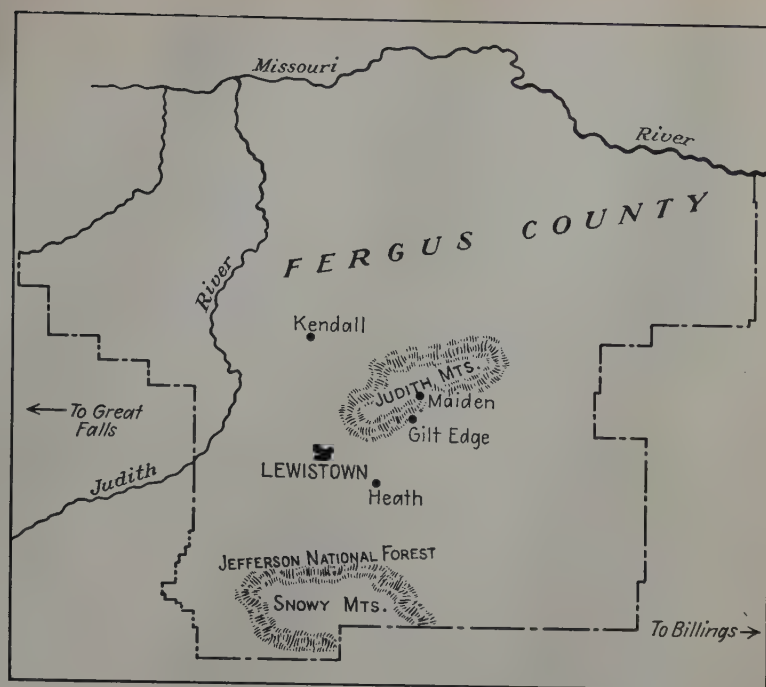


Fig. 1—Gold mine area of Fergus County, Montana.

by Gerald C. Mathis

FERGUS County, Mont., shown in Fig. 1, is known for its once famous gold mines near the old towns of Gilt Edge, Maiden, and Kendall. But at Heath, a small farming community near the foothills of the Snowy Mountains, thousands of tons of gypsum have been mined since 1916. The value of the gold taken from the mines of Fergus County runs into millions of dollars; the value of the products made from the gypsum of the Heath mine is also in the millions. The gypsum mine will continue to produce for many years, while the gold mines are closed today.

The Heath gypsum mine was developed by the Northwest Gypsum Co. in 1916. The products of the small mill built by this company were plaster, agricultural gypsum, and raw gypsum for cement manufacturing. The United States Gypsum Co. prospected the adjacent property in 1916 and 1917 by means of two adits and some exploration drill holes. During World War I, the company abandoned operations at Heath, but again became active in this area in 1928 when it purchased all the property of the Northwest Gypsum Co.

A long range program of expansion was begun with the building of a block plant and the purchase of additional land and mineral rights. A modern wall board plant was built in 1936. When the old wooden mill purchased from the Northwest Gypsum Co. burned in 1937, an all-metal calcining plant located at Blue Rapids, Kansas, was dismantled, moved to Heath, and erected on the site of the mill.

G. C. MATHIS is Mine Superintendent, United States Gypsum Co., Heath, Mont.

Discussion on this paper, TP 3484A, may be sent (2 copies) to AIME before April 30, 1953. Manuscript, Oct. 14, 1952. Los Angeles Meeting, February 1953.

In August 1951 a new mine was opened 3000 ft north of the mill. Rock is conveyed from the mine to the mill by an 18-in. belt conveyor.

The Heath gypsum deposit lies at the base of the Ellis formation of Jurassic Age. The Ellis at Heath lies unconformably on the black fissile shale of the Heath formation of the Big Snowy Group, which is Mississippian in age. The strike of the gypsum bed is $N45^{\circ}E$ and the dip approximately $3^{\circ}NW$; the regional strike and dip of the area.

The gypsum lies in a series of six layers separated from each other by thin seams of shale. The lower layer is composed of gypsum and shale intermingled and is not of commercial grade. The upper five layers are high in purity, but only the middle four layers are mined on the advance, as the top layer is left for roof support. This top layer is mined on retreating operations at the same time the pillars are robbed. Total thickness of the gypsum layers varies from 12 to 14 ft. Fig. 2 shows the average thickness and arrangement of the layers.

Mining Methods

The mine was originally developed by driving an adit into the gypsum horizon. The Northwest Gypsum Co. mined in the top four layers, the top layer being split, leaving about 12 in. for roof support. A modified room and pillar method was used: entries were 15 ft wide and rooms varied between 20 and 30 ft and were turned on 90° from the entries. Crosscuts between the rooms left pillars from 10 to 20 ft square. The rock was loaded by hand into 1-ton mine cars and pulled to a central gathering point by mules; from the gathering point, mules or horses were used to pull the string of cars to the mill tipple. This method was continued by the United States Gypsum Co. from 1928 to 1937.

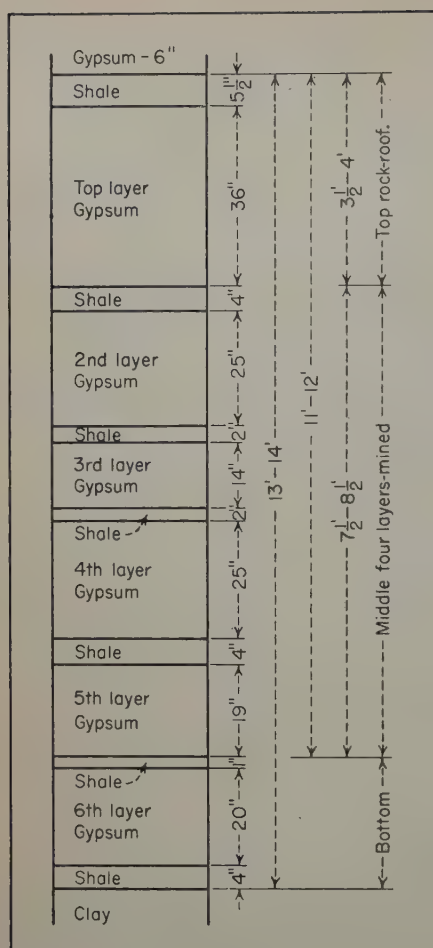


Fig. 2—Stratigraphic column, Heath gypsum deposit.

The addition of the wall board plant in 1936 placed greater demands on the mine, and a 4-ton Whitcombe electric locomotive was purchased in 1937. The locomotive was used to pull the strings of cars from the gathering point to the tippie, and the mules to pull cars to the gathering points.

In October 1942 a general earth movement in the vicinity caused the mine to cave in completely, and to meet demands for rock, a quarry was opened along the gypsum outcrop near the mill site. A stockpile of rock from one of the adits driven in 1916 on the adjoining property was loaded into trucks and hauled to the mill. A new mine to the north and east of the cave-in was laid out on a 45° herring-bone pattern, room and pillar method.

A portable loading slide with an Ingersoll-Rand triple-drum hoist for scraper loading was put into operation in 1943. At the same time a 6-ton Jeffrey electric locomotive was purchased for main line

haulage. The Whitcombe locomotive was used to switch the cars behind the scraper-loader. To meet the demand for more production during World War II, another scraper-loader unit and another 6-ton Jeffrey locomotive were purchased and put into operation in 1944. The scraper-loader method was successful but was limited in production and required considerable man power. The scraper-loader is shown in Fig. 3, the Jeffrey locomotive in Fig. 4.

Plans for further modernization of the mine were investigated thoroughly. In 1948 an 18HR-1 Joy Loader and two 60-E Joy shuttle cars were purchased, see Fig. 5. Plans as originally laid out utilized the mine cars for hauling rock to the tippie, but they were altered to permit the shuttle cars to haul direct from the face to the tippie. This required a new haulageway, which was driven with the Joy equipment, first utilized for production in 1949. One scraper-loader and the two Jeffrey locomotives were used as standby equipment in a reserve block along the rail haulage line. The Joy equipment used the 45° herringbone pattern, room and pillar method of the scraper-loaders.

Several factors influenced the decision to change to Joy equipment, flexibility being most important. The mine was nearly exhausted of face rock and robbing of roof and pillars was imminent. The saving in manpower was important also, as good steady workers were hard to find after World War II.

A new mine was opened in August 1951 on the property prospected in 1916. Only the Joy loader and Joy shuttle cars were moved from the old mine for loading and haulage in the new mine. At present there is no standby equipment in the new mine. Thorough inspection and prompt maintenance of the loader and shuttle cars is necessary to keep the equipment operating.

Table I offers a comparison of costs of the above mining methods, showing only those items concerned directly with loading and haulage. All figures are given in percentage of total costs, as labor and material prices have varied over the period covered by the table. The efficiency of the various methods is shown by the tons per total manhour.

Drilling and Blasting

At various times tests have been made to check the efficiency of different rounds in the gypsum at Heath. The best type to date is the Half-V round as shown in Fig. 6. A room round is drilled with 21 holes in vertical rows of 3 holes each. An entry round requires 18 holes. The rib holes of the half V are drilled 6 to 7 ft deep along one rib. The next row of holes is drilled on an angle so that the point of a 10-ft hole is at the point of the corresponding



Fig. 3—Portable loading slide equipped with a 20-hp Ingersoll-Rand triple drum hoist used in the Heath mine from 1943 to 1948.



Fig. 4—A 6-ton Jeffrey electric mine locomotive used for switching and main line haulage in the Heath mine 1943 to 1948.



Fig. 5—Joy 18 HR-1 loader and Joy shuttle car in the Shoemaker mine. This equipment was first used by U. S. Gypsum Co. in 1949.

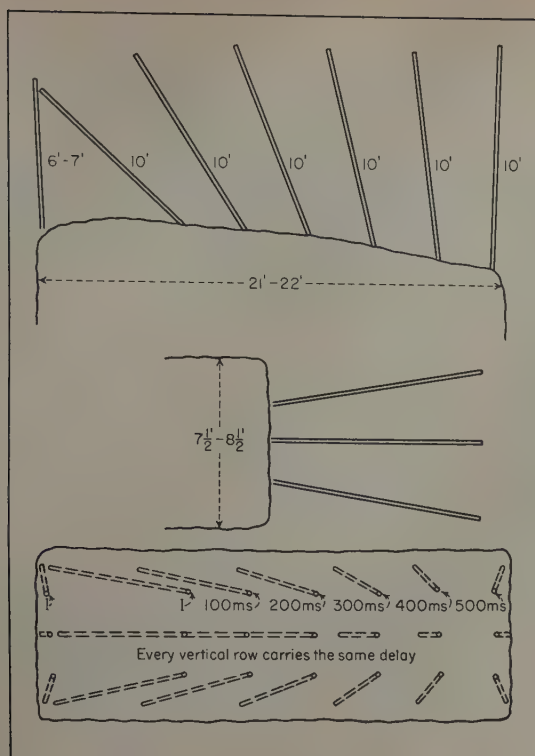


Fig. 6—Blast round, Shoemaker mine. From top to bottom: top view, side view, front view.

6 to 7-ft hole along the rib. Each succeeding row of holes is drilled on an angle leaving approximately 4½ ft of burden on the point of each hole. The last row of holes is along the rib opposite the V side.

The round is drilled with a Jeffrey A-6, electric post-mounted drill. One and one-half inch special twisted conveyor-type auger steel in 4, 6, and 10-ft lengths is used in conjunction with 1⅝-in. No. 37 coal cutter bits. The drill is mounted on a Jeffrey jack-post and has a small block at the top enabling the driller to raise or lower the drill to any desired height. The drill and post are shown in Fig. 7. The drillers each have a wheel-mounted cart which has a grinder mounted on it for sharpening the coal



Fig. 7—A Jeffrey A-6 post-mounted electric drill used for drilling gypsum in the Shoemaker mine.

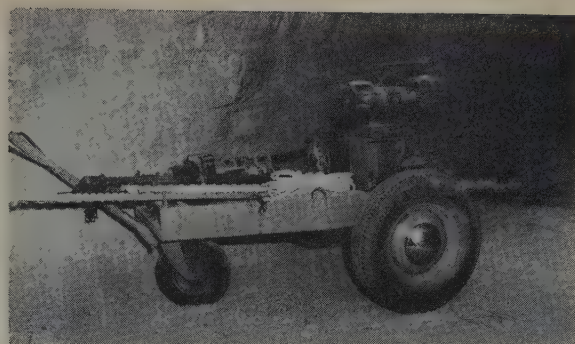


Fig. 8—Drill cart with grinder mounted used by drillers to move their equipment from face to face.

cutter bits. The cart, Fig. 8, with all the drilling equipment on it, is moved from face to face by hand, or in the case of severe grades by shuttle car. Plans are underway to make the drill carts self-propelled, thus eliminating lost production time on the shuttle cars.

The rounds are blasted with Du Pont Extra D 40 pct, 1⅝x8 in. dynamite. The number of sticks per hole depends on the depth and burden of each hole.

Table I. Direct Cost, Pct, By Years, No Depletion, Development, Or Depreciation Included

	1952 (6 Months) ¹ Pct	1950 ² Pct	1948 ³ Pct	1944 ⁴ Pct	1940 ⁵ Pct	1936 ⁶ Pct
Loading						
Labor	7.9	7.6	17.5	20.1	35.8	33.6
Repair labor	4.3	3.4	4.5	7.2	1.7	.9
Oilier inspector	1.8	1.6	3.9	2.5	1.9	1.9
Repair material	7.7	6.5	7.8	2.0	.5	3.2
Power	1.9	1.9	1.8	1.3	.7	2.7
Supplies and expense	6.3	4.3	2.9	10.1	3.7	11.9
Total	29.9	25.3	38.4	43.2	44.3	54.2
Haulage						
Labor	5.3	9.9	12.8	10.8	6.1	8.0
Repair labor	1.2	1.2	4.1	6.1	3.5	1.2
Repair material	5.4	3.2	2.3	.9	.4	.2
Supplies and expense	1.5	1.4	2.1	1.5	1.4	3.3
Power	1.8	2.3	1.5	1.3	.4	
Oilier inspector	1.1	1.3				
Total	16.3	19.3	22.8	20.6	11.8	12.7
Total Load and Haul	46.2	44.6	61.2	63.8	56.1	66.9
Tons Per Total Manhr	4.1	3.1	2.1	1.7	1.24	.9

¹ 1952: Joy loading and haulage, no standby equipment, Shoemaker mine.

² 1950: Joy loading and haulage, scraper-loader standby.

³ 1948: Scraper-loader and locomotive haulage, 2 units, last year.

⁴ 1944: Scraper-loader and locomotive haulage, 2 units, first year.

⁵ 1940: Hand loading, mule tramping, and locomotive main line haulage.

⁶ 1936: Hand loading, mule tramping, and haulage.

The round is primed with Du Pont millisecond caps in 100-m sec intervals. The decision to use millisecond caps was made in 1948, resulting in better fragmentation and more even round pull.

Shoemaker Development

Development of the Shoemaker mine was planned well in advance of the initial opening of the mine, which lies 3000 ft north of the mill on the property first prospected by the United States Gypsum Co. in 1916. Diamond drilling had proved that the deposit on the property was large and that it was an extension of the Heath deposit. The 1916 adit was dewatered and retimbered in 1947-48. One of the scraper-loaders and the 4-ton Whitcombe locomotive were moved to the Shoemaker property in 1949 and the work of completing the old adit in a horse-

shoe form was begun. However, work on the adit was slow because of the scarcity of labor, and the other end was not driven to the surface until 1950. This provided both adequate ventilation and two escapeways.

During the years 1947-50 a careful analysis was made of the various types of haulage systems to select the best one to get the rock from the Shoemaker mine to the mill. Although initial investment was high, the analysis favored belt conveying as the most economical and the most suitable to local conditions. The belt installation required a tunnel from the Shoemaker property to the mill. Part of the tunnel was cut 600 ft through the hill between the Shoemaker and Heath mines. This section is timbered with pre-cut creosoted timbers and is 6x6½ ft in the clear. In the Heath mine the tunnel was cut through rooms and pillars an additional 1400 ft to a point near the cave-in of 1942. Here a large room was made for a transfer point. From the transfer point the tunnel is timbered for 400 ft to the surface. The conveying system is housed in a corrugated sheet-iron shed from the surface to the mill, a distance of 400 ft. The 600-ft timbered tunnel with the belt conveyor installed is shown in Fig. 9.

A building to house the jaw crusher was built at the portal of the new mine. Across the coulee, 140 ft away, another building was built to house the hammer mill. The two are connected by a conveyor gallery which houses a 24-in. belt.

On August 4, 1951, the last rock was hauled from the Heath mine. The jaw crusher, hammer mill, and magnetic pulley were moved from the mill and re-installed in the buildings at the Shoemaker mine. The Joy loader was moved on a heavy-duty trailer and the shuttle cars were moved by bulldozer and truck. The rest of the mining equipment was moved by truck. During the installation of the crushing equipment, six rooms were started along one side of the adit for production. A trial run of 24 tons was made August 8, and after minor adjustments, production started August 9 from the Shoemaker mine.

Development of the Shoemaker mine has progressed steadily. Main entries were advanced 1200 ft north and 1200 ft northeast during the first year. A new system of main entries and sub entries was introduced with the opening of the new mine. Main entries are 16 ft wide with a 20-ft pillar between the parallel entries. Sub entries are turned on 60° angles from the main entries and are 18 ft wide and 400 ft long, with 20-ft pillars. Rooms are turned from the sub entries on 60° angles, and are 21 ft wide and 100 ft long with 15-ft pillars. Since panels are laid out as illustrated in Fig. 10, trolley

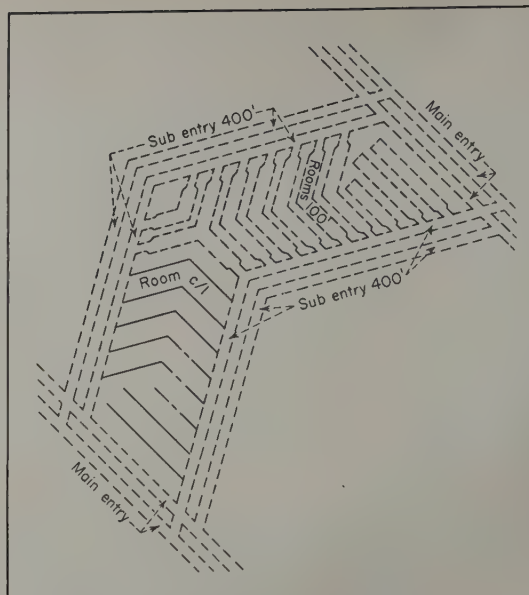


Fig. 10—Cross-cuts entries 120-ft c-l to c-l. No cross cut in rooms. Widths: main entries, 16 ft; sub entries, 18 ft; main and sub entry pillars, 20 ft; rooms, 21 ft; room pillars, 15 ft.

is needed only in the main entries. Shuttle cars have 600-ft cables on reels and can work all the rooms of a set of sub entries from the main entry trolley.

The drill pattern, shooting system, and method of loading and hauling used in the Heath mine were continued in the Shoemaker mine. The shuttle cars haul direct to the portal where the rock is dumped into a Joy elevator for elevating into the jaw crusher.

Rock as it comes from the mine is usually 18 in. or less. Any oversize is broken down with a Thor paving breaker before it is run through the jaw crusher. The jaw crusher, a Butterworth and Lowe, 18x30-in., crushes the rock to -6 in. at the rate of 50 tons per hr. A 24-in., 5-ply, nylon cord Good-year belt conveys the rock from the jaw crusher across a magnetic pulley to the hammer mill. This mill, a Jeffrey swing-hammer type, crushes rock to -2 in. at the rate of 70 to 80 tons per hr. A small 3-ton surge bin under the hammer mill feeds onto a Syntron feeder which can be regulated to feed the conveying system at any rate up to 100 tons per hr.

The conveyor belt, an 18-in. wide, 5-ply, nylon cord, mildew-resistant Goodyear, is in two flights. The first flight is 2000 ft long with a net rise of 28 ft. The belt is driven by a 20-hp, 440-v electric motor connected to a Falk gear reducer. From the transfer point there is a second flight which is 800 ft long with a net rise of 40 ft. This portion of the belt is driven by a 15-hp, 440-v electric motor connected to a Falk gear reducer. The carrier frame was fabricated by Haak Engineering, Denver, Colo. and is equipped with Rex-Stearns idlers. Self-aligning idlers are placed every 100 ft on the top side and every 250 ft on the return side. The entire system from the small belt over the storage bin at the mill to the jaw crusher at the mine is interlocked to protect all parts.

Acknowledgments

The author wishes to acknowledge the assistance given him by all members of the Heath plant and especially to William Freund, Mine Foreman, and O. M. Newsum, Mill Foreman.



Fig. 9—Part of the 600-ft timbered tunnel with conveyor system installed.

A Method for Concentration of North Carolina Spodumene Ores

by Mason K. Banks, William T. McDaniel, and Philip N. Sales

A process has been developed which produces spodumene concentrates assaying 6.0 pct Li_2O and 0.45 pct Fe_2O_3 , with 70 to 75 pct recovery of spodumene. Two flotation separations are required: the simultaneous removal of mica, feldspar, and quartz as a froth product using a cationic collector, and the subsequent removal of iron-bearing minerals as a froth product while spodumene is depressed as a tailing.

SINCE the close of World War II there have been numerous developments resulting in the increased use of lithium by industry. Notable among these has been the development of an industrial lubricant, based on lithium stearate, which is fluid at low temperatures, stable at high temperature, and insoluble in water. A new bleaching agent, lithium hypochlorite, has also been developed recently.¹ Other well-known uses are listed below:

1—Lithium oxide in ceramics reduces firing time, lowers melting temperatures, lowers thermal expansion, raises refractive index, and improves chemical resistivity. 2—Lithium carbonate and lithium strontium nitrate are used in pyrotechnics for their brilliant red color.² 3—Lithium chlorides and fluorides are used as fluxes for welding aluminum and magnesium.² 4—The affinity of lithium for nitrogen is the basis for a process used in purifying helium.² 5—Lithium chlorides and bromides are used in air conditioning for their powers to absorb organic amines, ammonia, and smoke.¹ 6—Lithium hydride has been used as a source of hydrogen to inflate life rafts and balloons used by the Air Forces and Navy.¹ 7—Lithium metal and lithium hydride are useful in the atomic energy program in connection with the production of the isotopes of hydrogen.³

During the 1942–45 period of World War II, Solvay Process Co. operated a flotation plant with a daily capacity of 300 tons of ore for the production of spodumene concentrate near Kings Mountain, North Carolina. The operation was shut down upon reduction of wartime demand for lithium chemicals, and

until recently there has been no production from the Carolinas.

In April 1950 an investigation of the problem of spodumene flotation from North Carolina ores was begun at North Carolina State College Minerals Research Laboratory at Asheville, N. C. This laboratory, operating under the combined auspices of North Carolina State College, North Carolina Department of Conservation and Development, and the Tennessee Valley Authority, undertook the spodumene project at the request of Foote Mineral Co. of Philadelphia. The work, involving a period of 15 months of intensive batch and pilot plant investigation, was completed in June 1951 and resulted in development of a process for concentration of spodumene by flotation which has been hereto unrecorded in the literature. A similar approach has been used before on iron, ilmenite, and other minerals, but the literature shows no record of its having been applied to spodumene. Parts of the process are now being used on a commercial scale at the Foote Mineral Co. plant near Kings Mountain, N. C., which has been in production since July 1951. It is the purpose of this paper to describe some of the results of the investigation.

The Orebodies

Spodumene-bearing pegmatites occur in a narrow belt 24.5 miles long and 1.8 miles wide extending southwestward from Lincolnton to Grover, N. C. According to statements judged to be conservative, the tin-spodumene belt of the Carolinas contains an estimated 4,300,000 tons of pegmatite ore averaging 15 pct spodumene. This amounts to "a reserve of at least 650,000 tons of spodumene (more than 20,000 tons of metallic lithium) to a depth of 100 ft."⁴ This figure includes only bodies having an average thickness of 35 ft and length of 550 ft. There are hundreds of pegmatites in the belt, many of which are less than 10 ft wide. The largest pegmatite in the area has a maximum thickness of 395 ft and maximum length of 3250 ft.⁴

These larger pegmatites are composed principally

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This report was prepared in cooperation with the North Carolina Department of Conservation and Development and the Tennessee Valley Authority. Discussion on this paper, TP 3488H, may be sent (2 copies) to AIME before April 30, 1953. Manuscript, September 17, 1952. Los Angeles Meeting, February 1953.

of albite and microcline feldspars, albite predominating, quartz, spodumene, and muscovite. There are many accessory minerals, but the important ones are cassiterite and beryl, and possibly columbite-tantalite. Approximate composition of ores from the belt which were used in this investigation is shown in Table I.

Table I shows that quartz and muscovite content remain fairly constant in all the ores, while feldspar content increases when spodumene content decreases, and vice versa. This is interesting because feldspar is potentially an important byproduct.

The Spodumene

According to Boyd,⁶ the following analysis is typical of the Carolina spodumene:

SiO ₂	Al ₂ O ₃	Li ₂ O	Fe ₂ O ₃	CaO	MgO	K ₂ O	Na ₂ O	Ignition Loss
63.84	28.61	7.26	0.19	0.04	Trace	Nil	Nil	0.16

Spectrographic analysis shows the presence of minor amounts of MnO, TiO₂, SnO₂, Ga₂O₃, Na₂O, and K₂O in the spodumene. One sample contained Rb₂O. Petrographic and X-ray studies indicated that these impurities are probably isomorphous replacements in the crystal structure of the spodumene.⁶

The molecular dispersion of the iron within the spodumene crystal has been substantiated by acid-leaching and magnetic separation tests in which the Fe₂O₃ content of spodumene crystals remained from 0.15 to 0.40 pct after magnetic separation and acid leaching.⁷

The iron content of the spodumene crystals is usually reflected in the color of the crystals, the colorless to white variety containing from 0.08 to 0.20 pct Fe₂O₃, the buff or cream variety containing 0.20 to 0.40 pct Fe₂O₃ and the yellow or greenish-yellow variety containing up to 0.65 pct Fe₂O₃. The darker the color the higher the iron content of the spodumene.

The amount of iron dispersed in the spodumene crystals is a significant factor, since it affects the use to which the concentrates may be put. For use in ceramics the spodumene concentrate should contain less than 0.5 pct Fe₂O₃, but for use in extraction of lithia for chemical purposes the presence of iron presents no special problem. None of the Carolina belt samples tested contained spodumene with low enough isomorphous iron content to yield a product with less than 0.2 pct Fe₂O₃. The average iron content of spodumene concentrates prepared from these ores was 0.40 to 0.50 pct.

The Carolina spodumene is somewhat variable in Li₂O content, but apparently averages approximately 7.0 pct or slightly in excess of this figure, although the textbook gives 8.03 pct theoretical Li₂O. Some variation is caused by weathering, for it has been established that Li₂O is lost by leaching when the rock undergoes weathering, just as the Na₂O and K₂O are lost in kaolinization of feldspar.

Review of Concentration Methods

The literature reveals the following methods used for production of spodumene concentrates: 1—The spodumene of the Black Hills of South Dakota was present in such large crystals that it was easily concentrated by selective mining and hand sorting. 2—The Bureau of Mines developed a method by which alpha spodumene was decrepitated at a temperature between 1000° and 1100°C, changing its form to beta spodumene, a soft pulverulent material, which

could be recovered from the harder gangue minerals by differential grinding following the decrepitation.⁸ 3—Froth flotation of spodumene using anionic collectors, or fatty acids, in an alkaline circuit was worked out by the Bureau of Mines and enlarged upon by Norman and Gieseke.⁷ This method was utilized by Solvay Process Co. at Kings Mountain, N. C., during its wartime operation. 4—Froth flotation with sulphonated oils in an acid circuit was also shown to be feasible, as well as a method of producing a feldspar-spodumene mixture by using cationic collectors in an HF circuit.⁷ 5—A mill at Keystone, S. D., operated by Lithium Corporation of America has been utilizing the heavy media method, treating 15 tons of -1½ in. ore per hr and producing a concentrate of 4 to 5 pct Li₂O.³

All the above methods have their advantages and disadvantages. The method described on the following pages is presented merely as a different approach and is acknowledged also to have both advantages and disadvantages.

The Flotation Process

The final process developed for spodumene concentration of North Carolina ores revolves around two primary flotation separations: 1—a process whereby mica, feldspar, and quartz are simultaneously removed as a froth product while spodumene and iron-bearing minerals are depressed as a tailing, and 2—the subsequent removal of iron-bearing minerals as a froth product while spodumene is depressed as a tailing. The removal of silicate gangue minerals in Step 1 is accomplished in an alkaline circuit, with a long-chain amine used as collector, and a colloidal

Table I. Ores Used in Investigation of Spodumene Flotation

Ore, Lab. No.	Sampled by	Spodumene, Pct	Feldspar, Pct	Quartz, Pct	Mica, Pct	Others, Pct**
808	Laboratory staff	38	30	26	3	2
837-A	Footo Mineral Co.*	36	33	26	3	2
971	Consolidated Feldspar Corp.	23	41	27	5	4
943	Footo Mineral Co.	19	45	25	4	7
969	Footo Mineral Co.	22	42	25	4	6
985	Footo Mineral Co.	13	56	22	4	5

* The Footo samples contained 0.03 to 0.05 pct Sn, or approximately 1.0 lb Sn per ton of ore. These came from the old Solvay quarry about 1 mile SW of Kings Mountain.

** Includes hornblende gneiss and sericite schist particles from wall rock and inclusions, as well as beryl, cassiterite, and numbers of accessory pegmatite minerals present in trace quantities.

depressant, such as starch or dextrine, used as selective depressant for the spodumene and for the iron minerals. The iron minerals are removed in an acid circuit, with sodium resinate used as collector and hydrofluoric acid as controller for prevention of spodumene flotation. Suitable frothers are used in both floats.

The gangue froth product containing mica, feldspar, and quartz may be split into its component parts for recovery of a feldspar byproduct. Since approximately one-third of the feed is generally recoverable as a feldspar concentrate, it was felt that efforts to recover it should be made. The general procedure for feldspar recovery would be to dewater the gangue froth product and then treat it in the conventional manner for feldspar ores, i.e., removal of mica with sulphuric acid and amine, washing by classification, then flotation of other iron-bearing silicates with sulphonated oil or fatty acid collectors, and finally flotation of feldspar from quartz with the conventional hydrofluoric acid-amine process.

Table II. Batch Test Results

Spodumene									
Lab. No. 808 Head Sample: 38 pct spodumene, 0.24 pct Fe ₂ O ₃ in spodumene crystals.									
Product	Wt, Pct	Pct	Units	Dist.	Fe ₂ O ₃	Al ₂ O ₃	SiO ₂	(1) Li ₂ O	(2) Li ₂ O
Spodumene conc.	29.4	91	26.8	70.8	0.26			6.55	
Iron froth product	3.2	62	2.0	5.3				4.85	
Mica froth product	4.2	4	0.2	0.5				0.45	
Feldspar froth product	21.0	14	2.9	7.7	0.07		67.0	1.25	
Quartz tailing	29.8	8	2.4	6.3	0.04		93.3	0.75	
Slime and oversize	12.5	29*	3.6*	9.5*					
Total	100.1		37.9	100.1					
Spodumene									
Lab. No. 837-A Head sample: 36 pct spodumene, 0.43 pct Fe ₂ O ₃ in spodumene crystals.									
Product	Wt, Pct	Pct	Units	Dist.	Fe ₂ O ₃	Al ₂ O ₃	SiO ₂	(1) Li ₂ O	(2) Li ₂ O
Spodumene conc.	33.8	89	30.1	83.6	0.44			6.50	6.44
Iron froth product	1.9	50	1.0	2.8	8.29			4.05	4.50
Mica froth product	1.5	3	0.1	0.3				0.35	0.52
Feldspar froth product	27.5	8	2.2	6.1	0.09	19.9	68.4	0.70	0.68
Quartz tailing	20.9	1	0.2	0.6	0.02		98.0	0.10	0.11
Slime oversize	14.4	18*	2.4*	6.7*					
Total	100.0		36.0	100.1					
Spodumene									
Lab. No. 943 Head sample: 19 pct spodumene.									
Product	Wt, Pct	Pct	Units	Dist.	Fe ₂ O ₃	Al ₂ O ₃	SiO ₂	(1) Li ₂ O	(2) Li ₂ O
Spodumene conc.	14.3	82	11.7	63	0.86			6.10	6.40
Iron froth product	2.6	55	1.4	7				4.35	4.00
Mica froth product	6.0	5	0.3	2				0.50	1.19
Feldspar froth product	37.9	7	2.6	14	0.26	20.6	66.0	0.70	1.40
Quartz tailing	20.1	1	0.2	1	0.14		94.1	0.10	0.36
Slime oversize	19.1	13*	2.4*	13*					
Total	100.0		18.6	100					
Spodumene									
Lab. No. 971 Head sample: 23 pct spodumene, 0.47 pct Fe ₂ O ₃ in spodumene crystals.									
Product	Wt, Pct	Pct	Units	Dist.	Fe ₂ O ₃	Al ₂ O ₃	SiO ₂	(1) Li ₂ O	(2) Li ₂ O
Spodumene conc.	22.8	84	19.2	83.5	0.56			6.25	
Iron froth product	1.0	94	0.9	3.9				6.70	
Mica froth product	4.4	1	0.0	0.0				0.10	
Feldspar froth product	36.6	4	1.5	6.5	0.12	20.4	68.1	0.40	
Quartz tailing	20.8	1	0.2	0.9	0.05		96.5	0.10	
Slime and oversize	14.3	8*	1.2*	5.2					
Total	99.9		23.0	100.0					

(1) By using curve showing Li₂O vs percent sinks in acetylene tetrabromide.

(2) By actual chemical analysis.

* Calculated using head sample analysis.

Distribution of spodumene based on percentage of sinks in acetylene tetrabromide.

Li₂O analyses by Foote Mineral Co.

The final process which breaks the ore into its component parts is described generally as follows:

1—Grinding. The ore as received was ground in a batch rod mill at 25 pct solids to pass a 28-mesh screen. The +28 mesh fraction, consisting almost entirely of mica, was removed. The -28 mesh fraction was deslimed by decanting over a 325-mesh screen. Approximately 0.5 lb caustic per ton of feed was used in the mill to keep the pulp dispersed for desliming. The oversize product consisted of approximately 1.0 pct weight of the feed, while slime loss came to 12 to 18 pct weight of the feed. Separations were obtained with feed as coarse as 20 mesh, and the general range of 20 to 28 mesh was satisfactory. The feed should be sized by screening or classification, with oversize returning to the mill in practice. The rod mill is preferred to the ball mill, because it produces a coarser, more granular feed with much closer size distribution. It is believed that ball milling would produce excessive slimes and a long particle size range, both of which have been shown to be undesirable in this process.

2—Rougher Gangue Flotation. The deslimed -28 mesh +325 mesh pulp was placed in a Denver 2000-g laboratory flotation machine and brought to 20 to 25 pct solids. The following reagents, calculated in pounds per ton of dry mill feed, were added: 1.0 lb per ton of lime, 0.5 of dextrine, 0.4 to 0.7 of Armac T, and 0.4 of B-23. The pulp was conditioned for 1 min, then floated for 4 min, with the quartz, feldspar, and mica being simultaneously removed as a froth product. The spodumene and iron-bearing

minerals were selectively depressed and removed as a tailing. Most of the froth was removed in 2 min. The float is heavy and fast if sufficient frother is used. The rougher is more selective if little or no frother is used, but requires more collector (Armac T) and an excessive float time. Lime and dextrine are more selective and cheaper than caustic and dextrine, although either may be used. If dextrine is not available, causticized-starch may be used, although dextrine is preferable. An excess of starch or dextrine does not break down the separation, but merely causes an increase in collector required. Approximately 0.5 lb per ton of dextrine or causticized-starch is generally sufficient to depress the spodumene. An excess of caustic or lime can cause complete breakdown of the separation as the amine collector is unstable when the pH is above 11.5.

3—Cleaner Gangue Flotation. The rougher gangue froth was next placed in the cell at 15 to 20 pct solids with 0.5 lb per ton of lime, 0.5 of dextrine, and 0.2 of B-23. It was conditioned 15 sec, then floated 4 min. This is virtually a repetition of the rougher, for the purpose of dropping out fine spodumene particles which were mechanically entrained in the heavy rougher froth. It requires no additional collector. If sufficient frother is used, the cleaner tailing or middling is a high-grade spodumene concentrate and may be combined with the rougher tailing.

4—Iron Mineral Flotation. The combined rougher tailing and cleaner tailing, consisting of a high-grade spodumene concentrate with substantially all the

iron-bearing minerals present, was then washed by decantation, thickened to 65 to 75 pct solids, and conditioned for 10 min with 0.35 lb per ton of HF (100 pct), 2.0 of sodium resinate, and 0.1 of B-24. The pulp was transferred to the cell and floated 4 min at 25 pct solids, with iron-bearing minerals being removed as a froth product, and relatively iron-free spodumene, except for molecularly contained iron discharged as a tailing. The tailing is the finished spodumene concentrate, which may be thickened, filtered, and dried in conventional manner. The sodium resinate collector was chosen over sulphonated oils as iron-mineral collector because it tends to concentrate the spodumene by floating mica and feldspar impurities along with the iron minerals, whereas the sulphonated oils tend to float pure spodumene along with the iron minerals. It is most important to maintain pulp density in the conditioner as high as possible before flotation of iron minerals.

5—Mica Flotation. The cleaner gangue froth was next conditioned for 3 min with 1.0 lb per ton of H_2SO_4 , 0.1 of B-23, and 0.05 of Armac T at 35 pct solids. Mica was then removed as a froth product.

6—Feldspar Flotation. The quartz-feldspar tailing from the mica float was washed by decantation, then conditioned 6 min at 50 pct solids with 1.5 lb per ton of HF (100 pct). It was then placed in the cell and diluted to 25 pct solids; 0.05 to 0.20 lb per ton of Armac T and 0.3 of B-23 were added before flotation. Feldspar was removed as a froth product and quartz was discharged as a tailing.

Results on batch testing several ores according to this outlined process are shown in Table II. Fig. 1 is a flowsheet of the process.

Discussion of Table II

Test results on four separate ores are shown in Table II to illustrate the amenability of various ore types and grades to separation by this method. Lithia content is given in two of the test summaries to show the comparison between actual chemical analysis for lithia and lithia content based on distribution of spodumene. It should be observed that lithia minerals must be considered on a dual basis: first, the mineral, which is all the mill operator can recover, and second, the lithia content, which is all-important to industry. Recoverable spodumene crystals in some instances may not contain more than 3.0 to 5.0 pct Li_2O owing to loss by leaching action of weathering, replacement of Li_2O by other molecules, etc. Other points which should be noted in explaining the seeming discrepancies between spodumene content and lithia analysis are: 1—The mica in the pegmatites contains a small amount of Li_2O . This throws any attempt to check spodumene distribution with Li_2O distribution slightly off. 2—The Li_2O content of the spodumene itself is variable. 3—Accuracy of Li_2O analysis is subject to serious error because of its complexity.

For these reasons, results were evaluated in terms of spodumene distribution rather than lithia distribution. Actual lithia analyses were made at intervals for check purposes and these are shown where available.

It will be noted that the iron content of the spodumene concentrates is usually slightly above that of the picked spodumene crystals from a given sample. The iron content of the crystals is part of the molecule and cannot be removed by physical means.

Ores 808 and 837-A were higher grade than average and in fresh, clean condition. No trouble was encountered in making very good lithia grades and

recoveries on this type of ore. Feldspar concentrates were also of good quality on these two ores.

Ore 971 was a medium-weathered, high-iron content spodumene ore, but was still definitely amenable to this process, as shown by the results.

Ore 973 was badly weathered and contained considerable kaolinized material and hornblende. This resulted in high-iron feldspar concentrates and high-iron spodumene concentrates. This ore severely tested the process, but spodumene concentrates containing 6.10 pct Li_2O minimum were prepared from it with a recovery of 63 pct. One of the chief conclusions drawn from handling ores like this was that an acceptable grade of concentrate could be made from badly weathered ores, but recovery would suffer. Hornblende was outstanding as the most difficult contaminant to cope with. It consistently reported with the spodumene concentrates when present in the ore, and was not easily separated therefrom. The chief loss of spodumene values occurred in the slimes, and the next highest loss in the feldspar concentrate. The spodumene recoveries shown compare favorably with those obtained in feldspar flotation, with which this type of processing is most closely comparable.

Pretreatment of Weathered Ores

It was found necessary to employ light attrition scrubbing on ore types which were weathered appreciably. Unless alteration products were removed from the mineral particle surfaces, selectivity was rather poor. This condition had been noted previously in application of anionic flotation to these same ores.⁷ The principal effect of weathering on this cationic float was to inhibit flotation of stained particles of quartz and increase amine consumption. This substantially lowered the grade of concentrates obtained. It was found that conditioning at 65 to 70 pct solids for 5 to 10 min with 4 to 5 lb per ton of caustic followed by a decantation wash was effective in removing the alteration products on particle surfaces. When the spodumene-gangue separation

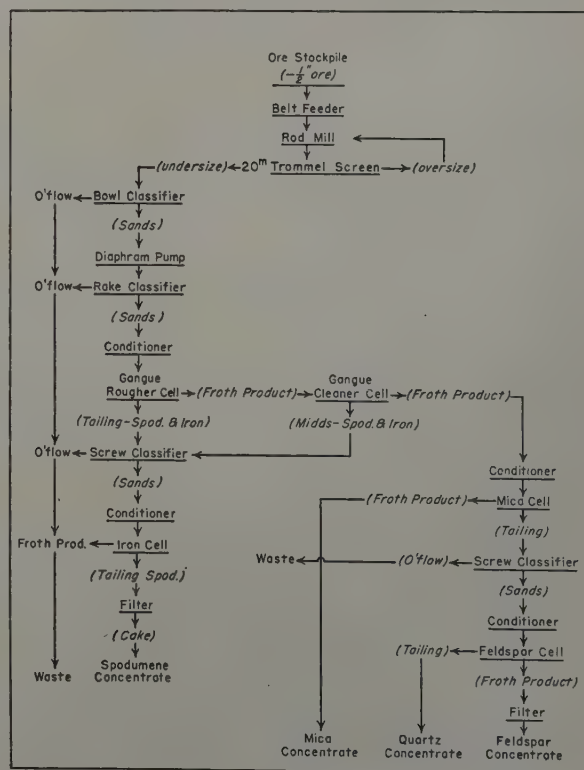


Fig. 1—Proposed spodumene flotation plant flowsheet.

Table III. Results of Batch Tests

Test 1 Dilute grind, 25 pct solids. No scrubbing.

Item	Wt, Pct	Spodumene, Pct	Spodumene, Units	Distribution of Spodumene
Spodumene conc	20.0	71.0	14.3	65
Gangue froth	69.3	8.0	5.3	24
Slime loss	10.7			11
Total	100.0			100

Test 2 Dilute grind, 25 pct solids. Scrubbed 6 min at 75 pct solids with 5.0 lb per ton NaOH. Washed.

Item	Wt, Pct	Spodumene, Pct	Spodumene, Units	Distribution of Spodumene
Spodumene conc	16.1	90.7	14.6	66
Gangue froth	72.9	5.4	3.9	18
Slime loss	11.0			16
Total	100.0			100

Test 3 Thick grind, 60 pct solids with 4.0 lb per ton NaOH in grind.

Item	Wt, Pct	Spodumene, Pct	Spodumene, Units	Distribution of Spodumene
Spodumene conc	20.2	86.3	17.3	79
Gangue froth	66.3	5.0	3.3	15
Slime loss	13.5			6
Total	100.0			100

was performed on weathered ores which had been scrubbed in this manner, selectivity was usually as good as that obtained on fresh, unweathered ore samples.

An alternate method was also found to be effective. This alternate method was to change the grinding pulp density from low to high percent solids, using 4 to 5 lb per ton of caustic in the mill, thereby getting a scrubbing action in the mill. This procedure resulted in improved flotation selectivity, but changed the nature of the mill discharge from a rather coarse, uniformly-sized feed, to a finer feed with a much greater range of size.

Of these two methods of handling weathered ores, the one utilizing dilute grinding conditions and a separate scrubbing step has been shown to be the best procedure, although the alternative use of the mill as a scrubbing device may be used if it becomes expedient to do so.

The following three batch test results, see Table III, indicate the beneficial effect of scrubbing on a weathered ore. In each case shown, treatment of the samples was identical except for grinding and scrubbing conditions as indicated. Feed was Lab. No. 969.

Pilot Plant Tests

Since the method outlined in the preceding pages represented a novel approach to spodumene flotation, it was decided to make pilot plant tests so that the spodumene-gangue and spodumene-iron separations could be studied under continuous operating conditions. Part of the necessary equipment was furnished the laboratory by Foote Mineral Co.

The flowsheet followed was the same as that shown in Fig. 1, except that the gangue froth product was collected as a bulk froth product with no attempt made to separate the mica, feldspar, and quartz. Grinding conditions were set up to utilize the mill as a scrubbing device, i.e., high density grinding with caustic. Feed rate for most of the tests was 300 lb per hr of $\frac{1}{2}$ in. ore, and a total of 15 tons of ore was used in these tests. Operating time on the test runs varied from 2 to 8 hr. Two ore samples were studied in this manner, Lab. No. 969 containing 22 pct spodumene, and Lab. No. 985 contain-

ing 12.5 pct spodumene. Seven complete tests were made on each ore. Variables studied in these 14 continuous pilot plant tests included amount of caustic used in the mill, pH studies in rougher and cleaner gangue floats, conditioning time and cell retention time in the gangue float section, comparison of pine oil, B-23, and cresylic acid as frothers, determination of optimum dextrine and amine amounts required, studies comparing sulphonated oils with soap as iron mineral collector, conditioning time and cell retention time in the iron float section. A short study was also made of the problem of concentration of the small amount of cassiterite which occurs in the spodumene ores. The cassiterite problem is not considered within the scope of this paper, but is mentioned to indicate that the presence of the mineral

Table IV. Results of Pilot Plant Tests

Operating Conditions

Test 2

Feed: $\frac{1}{2}$ in. Lab. No. 969, 22 pct spodumene, 0.89 pct Fe_2O_3 , fed at 300 lb per hr. Mill discharge - 28 mesh.

Equipment	Reagent	Quantity Reagents, Lb Per Ton	pH	Solids, Pct
Rod mill	NaOH	4.34		65
Rougher cell	NaOH	0.27	10.9	20
	Dextrine	1.15		
	Amine	0.90		
	Pine oil	1.13		
Cleaner cell	Dextrine	1.10	10.0	15
	Pine oil	0.69		
Fe conditioner	Sodium resinate	3.20	5.0	65
	HF	0.24		
	Pine oil	0.76		
Fe cell			5.4	20

Metallurgical Results

		Spodumene				(1)	(2)
Product	Wt, Pct	Pct	Units	Dist.	Fe ₂ O ₃ , Pct	Li ₂ O, Pct	Li ₂ O, Pct
Heads	100.0	22.0	22.0	100.0	0.89	1.90	1.30
Oversize	0.1	75.0	0.1	0.5		5.80	5.80
Bowl overflow	3.8	3.0	0.1	0.5	1.02	0.35	0.74
Rake overflow	3.2	3.0	0.1	0.5	0.68	0.35	0.87
Gangue froth	72.0	7.0	5.0	23.0	0.23	0.70	0.64
Iron froth	1.5	59.0	0.9	4.1	7.08	4.40	3.50
Spodumene conc	19.1	81.0	15.5	71.4	0.47	6.10	6.00
Total	99.7		21.7	100.0			

Operating Conditions

Test 14

Feed: $\frac{1}{2}$ in. Lab. No. 985, 12.5 pct spodumene, 0.35 pct Fe_2O_3 , fed at 325 lb per hr. Mill discharge - 28 mesh.

Equipment	Reagent	Quantity, Lb Per Ton	pH	Solids, Pct
Rod mill	NaOH	4.80		65
Rougher conditioner	NaOH	0.17	11.1	50
	Dextrine	0.98		
	Amine	0.68		
	Pine oil	0.90		
Rougher cell	NaOH	0.26	11.0	20
Cleaner cell	Dextrine	0.98	10.5	15
	Pine oil	0.46		
Iron conditioner	Sodium resinate	2.16		
	HF	0.70		65
	Cresylic acid	0.52		

Metallurgical Results

Product	Wt, Pct	Spodumene			Fe ₂ O ₃ , Pct	(1)	(2)
		Pct	Units	Dist.		Li ₂ O, Pct	Li ₂ O Pct
Heads	100.0	12.7	12.7	100.0	0.35	1.15	1.15
Oversize	0.1	80.0	0.1	0.7		6.05	6.00
Bowl overflow	7.7	5.0	0.4	2.8	1.64	0.50	0.87
Rake overflow	2.3	6.0	0.1	0.7	0.96	0.60	0.81
Screw overflow	0.4	12.0			1.00	1.10	1.80
Gangue froth	77.0	4.7	3.6	26.0	0.19	0.50	0.42
Iron froth	0.6	53.3	0.3	2.2	11.00	4.30	3.96
Spodumene conc	11.8	80.0	9.3	67.5	0.65	6.05	6.44
Total	99.9		13.8	99.9			

(1) By using curve showing percent Li_2O vs percent sinks in acetylene tetrabromide.

(2) By actual chemical analysis. Distribution based on percentage of sinks in acetylene tetrabromide. Chemical analyses by Foote Mineral Co.

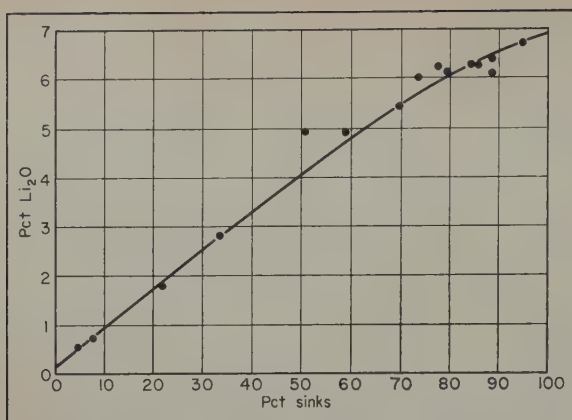


Fig. 2—Percentage of sinks in a given sample compared with percentage of Li_2O .

in interesting quantities was recognized and a method evolved to recover it.

Table IV shows the results obtained in pilot plant tests No. 2 and 14.

Heavy Liquid Control Method

The use of acetylene tetrabromide as a laboratory control aid is worthy of mention here in view of its extensive use in this investigation. The liquid has a specific gravity of 2.96. The only major constituent of the spodumene ores having a specific gravity higher than 2.96 was spodumene itself. It was found that —20 mesh ore samples, when placed in acetylene tetrabromide, resulted in a sinks product averaging 98 pct spodumene, the remaining 2 pct consisting of traces of cassiterite, horneblende, tourmaline, apatite, pyrite, and numerous others. Therefore, for all practical control purposes, the percentage of sinks in acetylene tetrabromide was the actual percentage of spodumene in the product.

A method was evolved in which the test samples were weighed, placed in acetylene tetrabromide contained in centrifuge tubes, and centrifuged for 10 min. Centrifuging brought about excellent separation of gangue minerals as floats and spodumene as sinks. Floats and sinks were then easily separated by a simple pouring technique, filtered, washed with benzene to remove excess tetrabromide, dried and weighed.

A graph, Fig. 2, was then drawn in which percentage of sinks in a given sample was plotted against percentage of Li_2O contained in the sample. Li_2O analyses for this graph were furnished by Foote Mineral Co. By obtaining the percentage of sinks in acetylene tetrabromide and reading the corresponding Li_2O value from the graph, it was possible to obtain a reasonably accurate Li_2O analysis.

This method was used as a control on laboratory batch tests and pilot plant tests. All products were quickly and easily evaluated from a lithia standpoint without the long and costly chemical analytical procedure for lithia determination. From 12 to 18 Li_2O determinations per 8-hr shift could be made by semi-skilled operators using this method. In addition to its use as laboratory control, the method could easily be adapted in commercial plant operating control.

Conclusions

The two flotation separations described in the foregoing pages produced spodumene concentrates from Kings Mountain ores on a continuous pilot plant scale. The spodumene concentrates contained an average of 6.0 pct Li_2O and 0.45 pct Fe_2O_3 with a

recovery of 70 pct available spodumene. If the iron removal step was not carried out the recovery of spodumene was approximately 75 pct. The process was successful on feed ground as coarse as 20 mesh. Weathered ores required light attrition scrubbing for satisfactory separation, but this was carried out in an acceptable manner by milling at high density with a high caustic addition.

The process is not recommended for use on ores containing less than 15 pct spodumene because recovery tends to drop off as spodumene content in the feed decreases. This is due to the fact that the gangue froth product inherently contains 3 to 5 pct spodumene, and this becomes an increasingly high percentage of the total distribution as the spodumene content of the ore decreases.

There were no particularly difficult operating problems. The two most important operating points, and the ones which might easily be overlooked, were proper pretreatment of weathered ores and strict maintenance of high pulp density in the conditioner before iron-mineral flotation.

The most difficult mineral to cope with was horneblende. Spodumene and horneblende are closely related mineralogically and exhibited similar reactions to the above described process. To remove horneblende satisfactorily, if it were encountered in appreciable amounts in the ore, it would probably be necessary to employ both flotation and magnetic separation.

Acknowledgments

The authors wish to acknowledge with sincere appreciation the assistance of the following persons and organizations in connection with this investigation: Dr. J. L. Stuckey, State Geologist of North Carolina, and his assistant, Sam D. Broadhurst; Charles E. Hunter, former TVA Geologist, for field sampling and geological assistance; Carolina Mineral Co.; United Feldspar and Mineral Co.; Foote Mineral Co., Research and Development Division, for furnishing checks on analytical control samples and part of the pilot plant equipment; W. H. Powell, R. B. Terrell, and W. R. Hudspeth for assistance in carrying out laboratory tests.

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Research on the Cutting Action of The Diamond Drill Bit

by E. P. Pfeider and Rolland L. Blake

IT is generally believed that the amount of diamond drilling will increase appreciably in the next decade, as the search for minerals throughout the world becomes more difficult and intense. An attendant problem may be one of short diamond supply, resulting in higher bit and drilling cost. With this background, the U. S. Bureau of Mines¹ and the School of Mines at the University of Minnesota² have established comprehensive research programs in diamond drilling. One of the several aims is the design of a more efficient bit, which would lower diamond consumption and increase rate of advance, both essential in reducing drilling costs.

The objective of the specific research problem³ discussed in this paper was an investigation of the cutting action of the diamonds set in a diamond drill bit, cutting action meaning the manner in which the diamonds cut or loosen the minerals in the rocks being drilled.

In the literature on cutting action such descriptive terms are used as: *grinding, wearing, cutting, breaking, shearing, scraping, melting, and chipping*. These actions were seldom described or defined. Grodzinski⁴ describes the cutting action of a single diamond in the shaping of certain types of material as "breaking out chips of the material." Brittle materials break as small separate chips, and tough materials, because of heat generated, give a continuous chip. Deeby⁵ said about diamond drills: "When diamonds are forced into the formation and rotated, they either break the bond holding the rock particles together, or they cause conchoidal fracture of the rock itself. The former action occurs when drilling in sandstones, siltstones, shales, etc. and the latter action when drilling in chert, flint, or quartz." He said that diamonds cut on the "grinding principle" but he does not define or elaborate on this action. The cutting action of diamonds on glass was first investigated about 1816 by Dr. W. H. Wollaston, an English physicist.⁶ The best glass-cutting diamonds have a natural or artificially rounded cutting edge. This edge first indents the glass and then slightly separates the particles, forming a shallow and nearly invisible fissure. Since none of the material is removed, this action is one of splitting rather than cutting.

No other reports of research work on the cutting action of the diamond were found, and further work was considered justified and advisable.

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Discussion on this paper, TP 3436A, may be sent (2 copies) to AIME before April 30, 1953. Manuscript, March 31, 1952. New York Meeting, February 1952.

It is impractical, even if possible, to observe directly the cutting action of a diamond drill bit in rock; therefore it was necessary to devise an indirect method. It was believed that a study of the following three observations would lead to a better understanding of the cutting action: 1—the appearance of the minerals or rock surface in the bottom of the hole, 2—the size, shape, and other characteristics of the drill cuttings, and 3—the condition of the diamonds in the bit.

The cutting action in a particular rock probably varies with bit pressure and speed. If the bit were slowly lifted off the rock, the effect of decreasing pressure might obliterate those bottom hole characteristics that are specific at the test pressure. Likewise, if the drill were stopped with the bit still in contact with the bottom of the hole, then decreasing speed effects would tend to obliterate the characteristics at the set test conditions. Therefore, in order to preserve those cutting effects impressed on the rock at test conditions, it seemed necessary to lift the bit off the bottom of the hole almost instantaneously once drilling conditions, i.e., revolutions per minute, pressure, and water flow became constant.

In addition to observing the cuttings, the bit, and the bottom of hole, it seemed desirable to collect some quantitative data for purposes of correlation with the observations and for a record of bit performance. Consequently such data as revolutions per minute, force applied, and rate of advance of the bit were recorded.

Six rock types, listed in Table I, were chosen for the tests. It was felt that these rocks had most of the variable characteristics of texture, bonding, and

Table I. Characteristics of the Rocks Tested

Rock	Source	Texture		Cementation		Minerals	
		Coarse	Fine	Good	Poor	Hard	Soft
Granite	Rockville, Minn.	X	—	X	—	X	X
Trap Rock	Dresser Junction	—	X	X	—	X	X
Taconite	Babbitt	X	—	X	—	X	X
Marble	—	—	X	X	—	—	X
Limestone	Bedford	X	—	X	—	—	X
Sandstone	Hinckley	X	—	—	X	X	—

mineral hardness met in the common rocks generally being drilled. The sandstone was so poorly cemented as to be friable, even though most of the cement was silica. The limestone, though well cemented, was quite porous.

Originally it was planned to conduct the test work with a full-scale drill unit, using EX bits, 7/8-in. core, 1 1/4-in. OD. The drill worked well, but was too cumbersome for rapid, accurate drilling of many short holes (1 1/2-in.) in varied rock types. A new

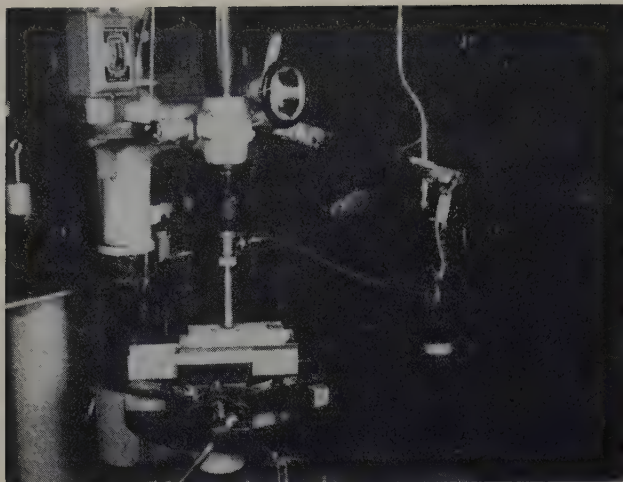


Fig. 1—Truco unit set up in drill press.

technique, employing a drill press and $\frac{5}{8}$ -in. Truco core bit, was developed to replace the larger drill, which was used thereafter only for check purposes.

The Truco Senior drilling unit was mounted in a drill press, see Fig. 1, having a $\frac{1}{2}$ -in. shank capacity and powered by a 0.94 hp electric motor through a gear transmission system and belt drive. The drill press could be varied between 80 and 305 rpm.

The two Truco F-1 bits used in these experiments were of 1-in. OD and $\frac{5}{8}$ -in. core diam. Both bits were surface-set, double roundnose, and had two waterways. The matrix was composed of sintered tungsten powder, and the diamonds were of the finest quality West African bortz, running approximately 100 per carat in size. A photo of the crown of one of these bits is shown in Fig. 2 and the pattern of their diamond settings is sketched in Fig. 3. The diamond wear during the course of the work was very slight, and bit condition is considered a constant for purposes of comparing the other factors.

The test blocks of the various rock types were approximately 3x3x8-in. in size and were drilled on four sides, see Fig. 4. They were placed in a galvanized tin trough, held in tightly by wooden wedges, and then clamped in the drill press benchvise. The tin trough caught the water and most of the cuttings. The overflow of water and fine cuttings was caught in a large container on the floor.

A binocular microscope was used to examine the cuttings, the drilling surface at the bottom of the holes, and the condition of the bit. The magnification commonly used was X21.2; consequently the depth of the holes was held to $\frac{1}{2}$ in. or less. A Leitz macro-camera was used for most of the photographs.

Bit rpm was one of the controlled variables. Tests were run at the minimum and maximum speeds of 80 and 305 rpm, and occasionally the intermediate setting of 170 rpm was used. It will be appreciated that equivalent linear speeds of 17 to 65 ft per min are much lower than the 300 to 450 ft per min obtained with an EX bit when rotating at the usual drilling speeds of 1000 to 1500 rpm. Such a variation in linear speeds may have considerable effect on the cutting action, although microscopic examination of particles caught during drilling at speeds of 80 and 305 rpm, respectively, showed little difference in size or shape of cuttings.

Force on the bit was the second controlled variable. The drill press feed was manual, activated by means of a lever arm, see Fig. 1. Constant, controlled force on the bit was obtained by hanging

weights on the lever arm and calibrating so that the thrust could be increased by increments of 25 lb. By using weights in this manner, the operator was able to lift the bit quickly off the bottom of the hole with the lever arm when operating conditions became constant. The drill press was calibrated for bit forces of 0 to 1000 lb when the arm was in its normal horizontal operating position.

Bit Performance

The bit performance curves drawn from the test data obtained in the drilling of the six rock types are shown in Figs. 5 and 6. These curves do not express the cutting action directly. However, they do permit comparison of the cutting efficiency for the same bit in different rocks, and, of greater importance, represent the measurable effect of various phenomena noted in the study of cuttings and of the bottom of the hole. A discussion of the more significant results follows.

Granite: The drilling rate increased almost proportionately to changes in both force on bit and rpm of bit. The drill stalled at 550 lb and 305 rpm, necessitating a reduction to 170 rpm to attain bit forces up to 1000 lb. Had it been possible to test at speeds up to 1000 rpm, it seems unlikely that the increase in rate of advance would have held proportional to increased speed of rotation. In other words, rate of advance vs rpm curve would have changed from a straight line relationship into an ever flattening curve. The explanation for this probably lies in the fact that the cuttings are ground increasingly finer before being washed away from the faster moving diamonds; hence the drilling efficiency decreases.

Trap Rock: The advance in trap rock at 80 rpm was similar to that in granite and taconite. Drilling was discontinued at 700 lb because most of the spaces between the diamonds on the bit crown were plugged, and the bit speed and drilling rate were reduced. The drilling rate at 305 rpm again dropped off because of this plugging of the bit. At 500 lb the bit nearly stopped rotating. It was cleaned and temporarily gave good results, but at 600 and 650 lb plugging again lowered the rate of advance.

Taconite: The drilling rate in taconite at 80 rpm showed an increase proportional to bit force, and

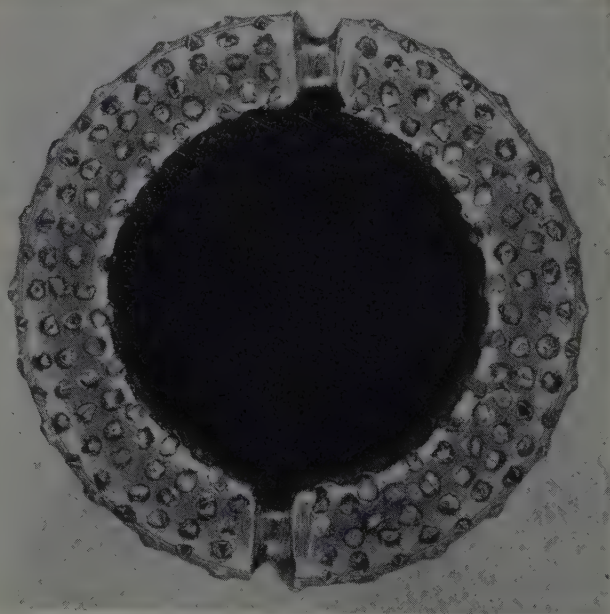


Fig. 2—Truco bit F-1. About three times actual size.

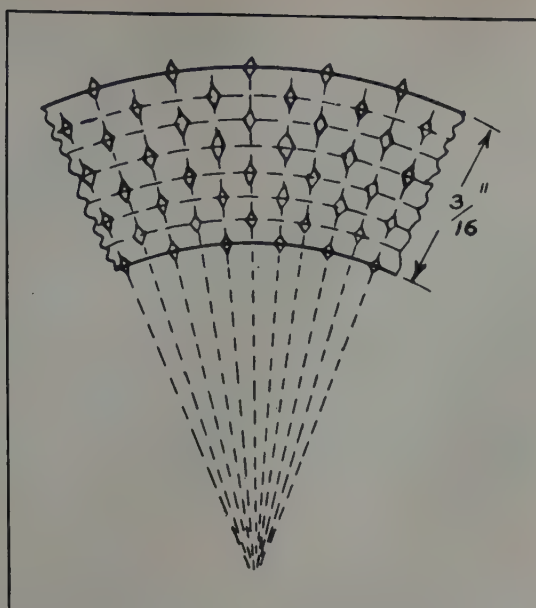


Fig. 3—Diamond arrangement in Truco F-1 bit.

gave a curve very similar to that observed in drilling granite. However, the rate of advance at 305 rpm did not increase in a direct ratio to the increase in bit speed from 80 rpm, as it did in drilling of granite. The reason for the disproportionate advance at the higher speed, 305 rpm, as compared with the rate at 80 rpm, cannot be explained by the facts at hand. There was no tendency for the bit to plug, and it was possible to drill at conditions up to 900-lb bit force, giving a penetration of $2\frac{5}{8}$ in. per min, before the revolutions per minute slowed down appreciably.

Marble: The bit performance in marble was very poor considering the fact that marble is composed of recrystallized calcite and/or dolomite, both of which are much softer than the quartz and feldspar in granite. Furthermore, deviations from a smooth curve were greater for drilling in marble than for drilling in granite. These results may be explained by the tendency of the cuttings to cling to the crown of the bit and to build up a cake on the bit crown. The word *cake* in this paper will refer to the fine powder of a mineral, s, that agglomerates and covers the grooves and/or ridges and may cling to the matrix of the bit, thus plugging the spaces between the diamonds. The bit was cleaned whenever this



Fig. 4—Arrangement of holes on one side of a columnar block of granite. Holes are about $\frac{1}{4}$ in. deep.

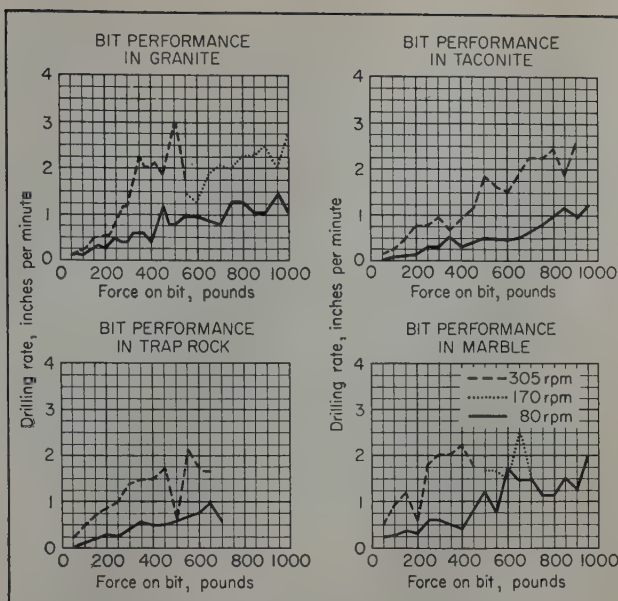


Fig. 5—Bit performance for Truco F-1 bit in drill press set-up.

condition was noted, but it was not possible to prevent this caking during the running of individual tests, even though the water flow was considerably increased. Warm water was tried, but no consistently improved results were obtained. The maximum drilling rate before stalling was about $2\frac{1}{4}$ in. per min at both 170 and 305 rpm, as compared to about 3 in. per min for granite.

Limestone: The drilling rate in limestone compared favorably with that in sandstone at low bit forces. At bit forces exceeding 250 to 300 lb, spaces between the diamonds on the bit crown became filled with mashed cuttings, and there was an involuntary reduction in revolutions per minute. As with marble, the test results were erratic and were further evidence that the chief cause of poor performance for this particular bit in soft rocks is the caking of the bit crown by the cuttings. This caking did not occur in drilling granite or sandstone. Force on the bit seemed to be the principal cause of the caking action in limestone, there being a tendency to mash the cuttings into a cake at the critical force of about 200 lb. Here again a generous flow of warm water was used in an attempt to eliminate the caking action. The bit performance curve for 80 rpm indicates that warm water may have relieved the situation during drilling at 250-lb force, but not at a force of 300 lb.

Sandstone: Drilling rates in sandstone, as in granite, were roughly proportional to increases in bit force and revolutions per minute when within the power limits of the drill unit. There was no tendency of the bit to plug by caking, and maximum rate of advance appeared to be about 9 in. per min, or three times the rate obtained in granite under similar conditions.

In general, advance for this particular Truco bit design in the various rock types appears to be roughly proportional to revolutions per minute and bit force, as long as the water can carry the cuttings away from the bit crown rapidly enough to prevent regrinding or plugging of the spaces between the diamonds. Plugging occurred in the finer grained rocks, marble, limestone, and trap rock, particularly if soft minerals were present. It was evident especially at the higher bit speeds. There appears to be a rela-

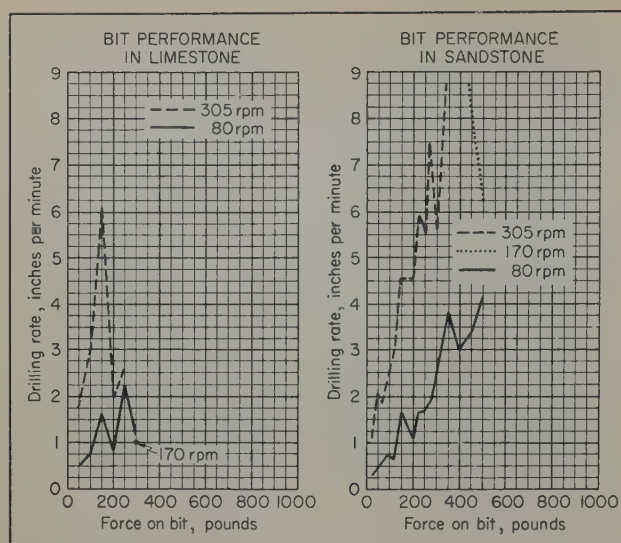


Fig. 6—Bit performance of Truco F-1 bit in drill press set-up.

tionship between plugging and a critical force on the bit, ranging from 200 lb for marble and limestone to 350 lb for trap rock.

Examination of Bottom of Hole

Most striking under microscopic examination of the annular surface at the bottom of the holes, see Figs. 7a through 12, were the concentric, alternating ridges and grooves in the rock. They were usually well defined at slower bit speeds and forces but gradually became indistinguishable when bit speeds and pressures increased or when the bit became plugged. Another characteristic was the frosted or sugary appearance of the annular surfaces. This became more pronounced at higher bit forces, especially in granite. *Frosting*, in this sense, is believed to be caused by the refraction and reflection of light from irregular surfaces of the minerals, as well as from minute fractures beneath the exposed surfaces of the minerals.

Granite: In an examination of the newly-cut surfaces of the holes drilled in granite, see Figs. 7a and 7b, the quartz appeared conchoidally fractured, with a frosted appearance. The quartz seldom stood in ridges, which were broken off. Minute fractures extended in all directions deep into the mineral. Very little separation occurred along these fractures. Thus free particles were not formed deep in the minerals

ahead of the diamonds. This fracturing might be likened to that which occurs when an ice-pick splits the ice ahead of it without separating the ice into pieces. It is also very similar to the fissures noticed by Wollaston in cutting glass with diamonds. As would be expected, the fracturing appears to extend deeper at higher bit forces.

Feldspar also was fractured or shattered beneath the mineral surfaces, though not as pronouncedly as was the quartz. It often broke along cleavage directions. Feldspar stood in ridges better than quartz, probably because it was not as brittle or hard. If the cleavages of the feldspar were nearly perpendicular to the direction of bit advance, these ridges were often sheared off low, along the cleavage.

Biotite usually stood in good ridges and did not appear fractured in depth. Ordinarily the flakes separated along their basal cleavage. The biotite was cut clearly when at right angles to the cleavage, especially in the grooves. Ridges in which the cleavage was vertical showed smoothly cut edges. Those with inclined cleavage had ragged edges and often had minute *books* of mica plucked out. Biotite was the first mineral to show a mashed effect, the finely ground particles, slightly moistened with water, being sufficiently cohesive to be pressed into a cake. At 80 rpm and 475-lb bit force some of the biotite in the grooves did not break away, but became crushed and mashed into a cake beneath the diamond points. However, this caking was never sufficient to plug the bit or appreciably affect its drilling action. At all bit forces greater than 475 lb some biotite was mashed. Quartz and feldspar did not appear as a mashed cake on the bottom of the hole at any bit forces.

The cutting action in granite, then, seems to be a shattering at depth of the hard and brittle minerals, such as quartz and feldspar, forming a network of minute fractures. This splitting often occurs along cleavage planes, as in feldspar. The fractures aid the succeeding diamonds in breaking out particles toward the free faces, or exposed surfaces. A soft, platy mineral, such as mica, is separated along its cleavage and is broken at right angles to this cleavage. Many of the mineral particles are ground or broken finer before they are washed free from the action of the bit. Such fine particles from minerals like feldspar may be mashed into a flake, whereas those from the soft minerals like mica are mashed into a cake. The word *flake* in this report will refer

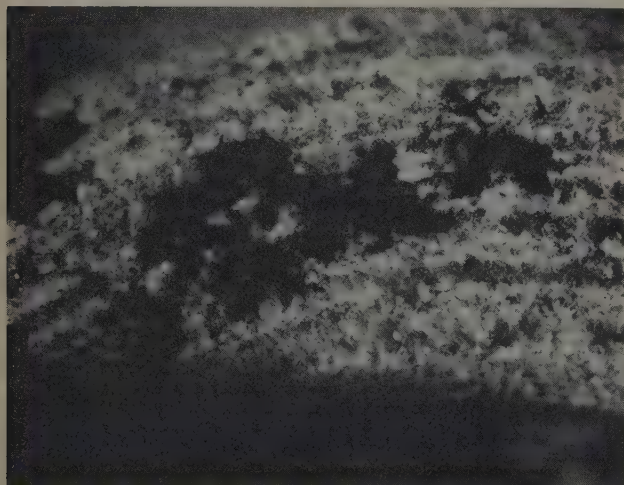


Fig. 7a—Bottom of hole in granite, X12, 80 rpm, 500-lb force, 3/4 in. per min.



Fig. 7b—Bottom of hole in granite, X12, 305 rpm, 100-lb force, 1/5 in. per min.

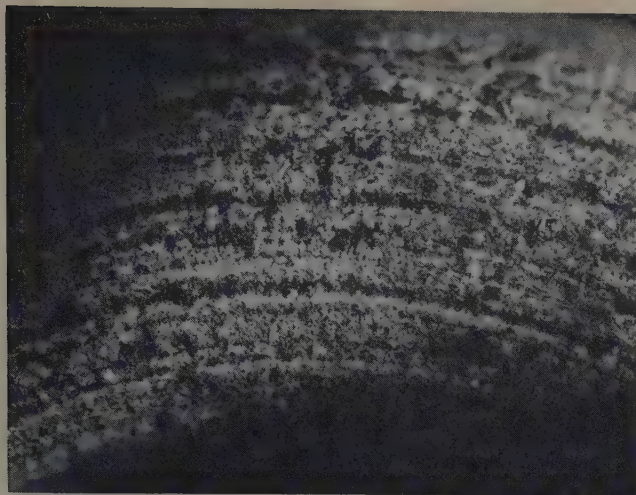


Fig. 8—Bottom of hole in trap rock, X12, 305 rpm, 150-lb force, 3/4 in. per min.

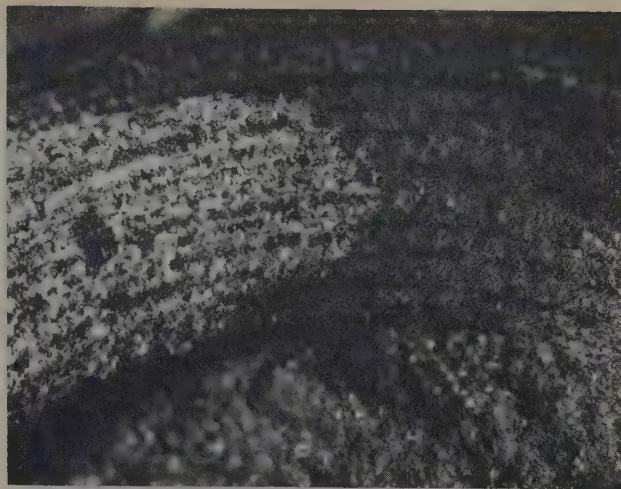


Fig. 9—Bottom of hole in taconite, X12, 305 rpm, 850-lb force, 1 7/8 in. per min.

to rock flour or mineral, s, that has been mechanically mashed into a thin, slightly curled chip. Cake plugs the bit when in sufficient quantity, and slows or stops the drill; flakes apparently have only a slight effect on cutting efficiency.

The bottoms of some 69 holes in granite were compared to determine what visible effect various bit speeds and forces have on the cutting action. These effects are summarized in Table II.

Trap Rock: Grooves and ridges usually were fair to well-developed in the holes drilled in trap rock, as evident in Fig. 8. A dark platy mineral, like biotite, broke along a basal cleavage, as did the biotite in granite. Soft silicates mashed and formed cake, which often plugged the bit. This mashing and caking was not so pronounced as that in marble

and crumbles beneath the diamonds. Fracturing was very shallow in magnetite. It did not form a cake, presumably because of its hardness and tendency to crumble rather than to mash.

Silicates in the taconite showed a deeply fractured appearance similar to that of quartz and feldspars in granite. Ridges were only slightly fractured. Both grooves and ridges were fairly well-developed at lower bit forces but became indefinite at higher bit forces. There was a slight tendency for some of the soft silicates to form flakes, but not to cake.

In areas where silicates and magnetite were intimately mixed, each mineral cut as described above, but ridges and grooves usually were well-developed. Although taconite is a tough rock to drill, this quality appears to be imparted more by the silicates than by the magnetite.

Marble: A photograph of the bottom of a hole drilled in marble is shown in Fig. 10. Grooves were found to be shallow, and ridges were low in all holes. When visible, grooves always had a frosted appearance from fragments fractured but not completely broken free. Ridges were usually frosted, but occasionally revealed the dark, very fine, recrystallized texture of the marble.

Fracturing of the marble did not extend deeply or vertically, as in the quartz and feldspar, but was shallow and gently inclined. Grooves were often filled with the cake mentioned previously. It had a white, chalky appearance, and in several holes was thick enough to cover both ridges and grooves with a smooth layer. Tiny traces of the diamond points in this cake suggest that the bit also was plugged with cake, riding high on the contact of both cake accumulations and with only the tip of the diamonds scratching the cake on the bottom of the hole. The low drilling rates in marble support this conclusion.

It is believed that flocculation of the cuttings beneath the bit may have aided the caking effect. When a 0.1 pct, by weight, quebracha solution was used as a dispersant, the caking effect was low to medium in comparison with previous holes, and drilling results suggest that this dispersing agent prevented some caking.

The cutting action in marble seemed to be a shallow shattering of the mineral particles, producing very fine angular fragments easily mashed into a cake, and even into large flakes, some of which

Table II. Comparison of Effects on Cutting Action of the Diamond Bit on Granite as Caused by Increase of Bit Speed, RPM, and Bit Force

Increased Bit Speed	Increased Bit Force
1—Greater impact of diamonds possibly resulting in increased depth and intensity of fracturing.	1—Greater compression by diamonds resulting in increased depth and intensity of fracturing.
2—No increase in the number of fine particles remaining attached to the rock surface.	2—Increase in the number of fine particles remaining attached to the rock surface, resulting in more frosted appearance.
3—No further penetration of diamonds into the rock, but rather the removal of more cuttings per unit of time because of increased linear speed.	3—Deeper penetration of rock by diamonds, cutting out more and/or larger particles per unit of time.
4—At low bit forces, higher bit speeds resulting in smoother mineral surfaces. Less noticeable effect at high bit forces.	4—At low bit forces, higher bit speeds resulting in smoother mineral surfaces. Less noticeable effect at high bit forces.
5—No effect on ridges.	5—Lower and narrower ridges.

and limestone. Hard silicates again fractured ahead into the minerals, but not so deeply as in quartz. In general the holes drilled in trap rock looked much like those drilled in granite.

Taconite: The bottom of a hole drilled in taconite is shown in Fig. 9. The dark mineral is magnetite, and the highly reflective area is composed of silicates. Magnetite showed an irregular or hackly fracture and a tendency to break out in tabular chips. Its grooves and ridges were poorly developed, probably because this brittle mineral fractures easily

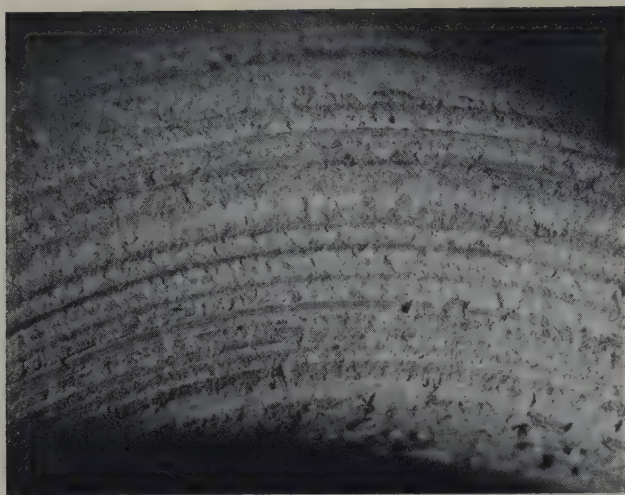


Fig. 10—Bottom of hole in marble, X12, 305 rpm, 150-lb force, 7/16 in. per min.

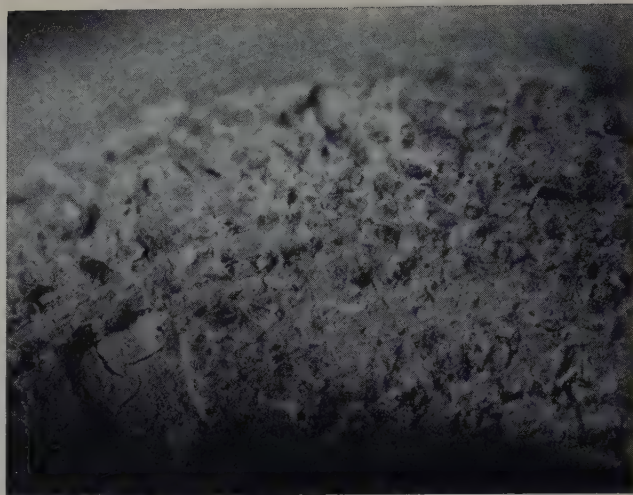


Fig. 11—Bottom of hole in limestone, X12, 305 rpm, 200-lb force, 1 7/8 in. per min.

were observed resting on the bottom of the hole. As bit force increased, the diamond points penetrated further into the soft calcite and accentuated the tendency to cake, 1—by cutting increasingly greater amounts of calcite and 2—by decreasing the clearance space between diamonds so that the wash water could not remove the soft particles before they become mashed between the bit and the bottom of the hole.

Limestone: Fig. 11 is a photograph of the bottom of a hole drilled in the oolitic limestone. Grooves and ridges varied in condition from fair to entirely missing. Cake is abundant, as it was in the marble, and a large thick piece may be seen in the lower left corner of the photograph.

The cutting action in oolitic limestone is a very shallow fracturing of the grains and their cement. They are so soft that they yield easily and do not fracture deeply. A massive limestone probably would be cut more like a marble, wherein the mineral grains are fractured evenly rather than being plucked out. The degree and manner of caking seems about the same in limestone and marble.

Sandstone: Sandstone showed no grooves and ridges, see Fig. 12. Two factors contributed to this: 1—the diamonds did not penetrate quartz grains very deeply, but seemed to ride on the diamond points, and 2—the poorly cemented grains were easily separated. Trails of finely powdered quartz occurred where the diamond points rode on the quartz. The dark spots in the photographs are shadows in the many cavities. Some of these cavities were original pores not filled by the cementing media; others were formed by the plucking out of grains or aggregates of grains.

Much of the drilling broke the quartz grains loose by shattering their cement, but over half the grains were broken. As the bit force increased, the percentage of quartz grains broken rose from a low of 50 pct to a constant of 75 to 80 pct at forces of 100 lb and greater.

The cutting action in sandstone was a deep fracturing of the quartz grains, like that occurring in the quartz of a granite. This fracturing broke the quartz grains if the silica cement held them tightly, or it broke the cement itself. Some of the grains and cement were pushed forward into cavities, where they were broken up further by the bit. This porous texture presents many free faces to which the fractured quartz could be broken, and its effect

on the cutting efficiency is evidenced strikingly in the bit performance results.

From a study of each of the six types of rocks drilled, it is apparent that each mineral is cut in a fairly definite manner, depending upon its hardness, toughness or fibrous structure, cleavage, tenacity, and probably several other physical properties. The cutting action in rocks is less definite than that in minerals because rocks are usually heterogeneous aggregates of different minerals. The cutting action thus becomes modified by variations in mineral composition, texture, porosity, and degree and type of cementation.

Microscopic study and screen analysis of the drill cuttings aided and confirmed the interpretation of the cutting action as described above.

Shape of Cuttings

Granite: Quartz cuttings were angular, conchoidally fractured fragments, seldom internally shattered. Feldspar presented an irregular broken appearance. When cleavage was visible, the fragments were tabular with jagged edges. The feldspar produced some thin curled flakes at higher bit forces, presumably owing to a mashing action. Biotite cuttings were irregularly edged sheets of varying thickness, from those nearly transparent to opaque books about 0.5 mm thick. Some of the thin sheets were bent, but very few flakes of biotite were formed by a mashing action.

Trap Rock: The cuttings of trap rock contained numerous light greenish-gray fragments of frosted appearance, probably silicates. There were also many thick, curled, mottled black and green flakes. Although the fine rock flour mashed into a thick cake in the bottom of the hole, it did not collect into aggregate lumps as it did in the cuttings of marble and limestone.

Taconite: Magnetite occurred as flat, hackly fractured chips, and occasionally as very tiny flakes. There were some angular quartz fragments and many green and colorless silicate fragments, about half of which had been mashed into flakes.

Marble: Marble cuttings consisted of a few fragments, up to 0.5 mm in size, but mostly of fine rock flour. The particles often clung together, even in a dried state, to form aggregates. Perhaps these aggregates represented part of the cake that continuously filled the grooves in the bottom of the hole. There were many curled, shiny flakes, as shown in Fig. 13.

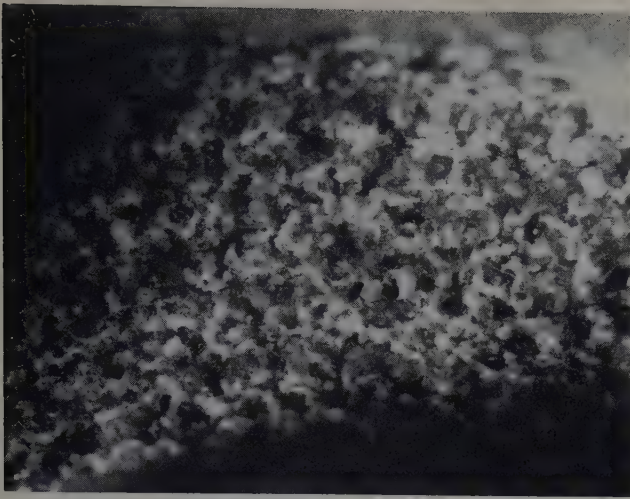


Fig. 12—Bottom of hole in sandstone, X12, 80 rpm, 75-lb force, 7/16 in. per min.

Limestone: Oolitic limestone cuttings contained buff-colored colites; rock flour, and broken oolite fragments. Some oolites were still cemented in aggregates. The fine particles aggregated into lumps, thereby giving the impression that a coagulant had been used.

Sandstone: Sandstone cuttings showed very jagged conchoidally fractured pieces of quartz up to 1 mm in diam. Crystal faces were visible on some fragments. Cuttings resembled many tiny chips of glass.

Size of Cuttings

The consensus of opinion in the diamond drilling profession is that a relationship exists between speed of drilling advance and size of cuttings. Samples were caught during nine of the tests drilled by the Truco bit, 1-in. OD, and 14 test runs in granite with an EX bit, 1.5-in. OD. These samples were sized by screening, and some of the results are presented in Table III.

The following observations are made on the basis of this preliminary work, although much more research should be done on this phase of the problem in order to develop definite conclusions.

1—A rough correlation exists between rate of advance and size range of cuttings, i.e., the higher the rate of advance, the coarser the particles. The results obtained in sandstone are particularly significant in this respect.

This relationship is similar to that found by Ertl and Burgh in their test drilling of the oil shales at Rifle, Colo., with percussion drills. Their conclusion was that "the speed of percussion drilling varies directly with the coarseness of cuttings produced."⁶ They further stated that rock-drilling speeds could be increased by improving the design of rock bits to produce coarser cuttings and by removing the coarse cuttings as rapidly as produced. Much the same statement can be made of diamond bits and drills.

2—During drilling in granite, more fine particles, —270 mesh, were formed at the higher bit speeds and the lower bit forces, as would be expected.

3—The average weight percentage of —270 mesh, 0.053 mm, cuttings from the Truco bit in all six rock types was 63.7 pct. Cuttings from granite ranged from 65 to 78 pct —270 mesh.

4—Cuttings from granite ranged from 79 to 88 pct —270 mesh when drilled with the standard EX double roundnose bit having diamonds running 32

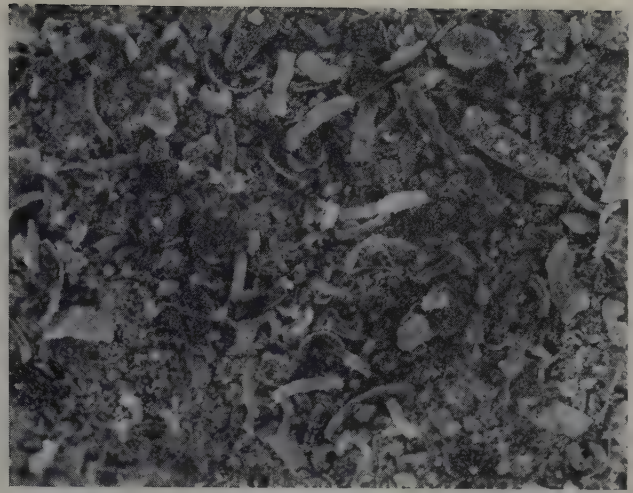


Fig. 13—Cuttings from marble, X12, 80 rpm, 100-lb force, 1/4 in. per min.

stones per carat. Surprisingly, these figures are 10 pct higher, for the finest fraction, than those produced by the Truco bit, which has much smaller stones. However, the EX bit was rotating at speeds of 800 to 1200 rpm, in contrast to 80 and 305 rpm for the Truco bit. The finer particles produced by the EX bit are probably accounted for by this higher bit speed, together with the fact that there are many more individual diamonds in the EX bit than in the Truco bit, subject to a lower unit force.

Interpretation of Cutting Action

Certain concepts of the cutting action of the diamond drill bit have been developed from the interpretation of the research work reported herein. A discussion of these concepts follows.

Breaking of Rocks and Minerals: Inasmuch as most rocks are composed of mineral particles of varying size, shape, and composition, rocks are essentially heterogeneous. They will break more easily along bedding, lamination, and grain boundaries than across grains or perpendicular to these directions.

Each mineral particle of a rock, however, is a homogeneous substance and will break according to its internal physical characteristics. Dana states that the crushing, or breaking, characteristics of minerals depend upon cohesion and elasticity.⁷ These characteristics, in turn, are exhibited in such physical properties as hardness, tenacity, brittleness, fracture, and cleavage. Cohesion represents the force of attraction between molecules of a given substance; elasticity is the force tending to restore the molecules under strain to their original positions. Both these internal forces must be overcome when minerals in a rock are broken into fragments.

In diamond drilling, the force on the bit causes diamonds to penetrate the rock, and minerals, by a compressive force. This force sets up internal stresses. In brittle minerals or rocks, these stresses will not be relieved readily by strain, that is, deformation, but rather by the formation of tension or shear fractures.

The rotary bit speed causes two actions by the diamonds on the rock or minerals: 1—an impact, and 2—a shearing force. The first stage of an elastic impact is compression; the second stage an elastic restoring force. Both stages may be relieved by the development of tension and/or shear fractures. The shearing force, likewise, may cause these fractures.

Breaking to Free Faces: In drilling rocks with

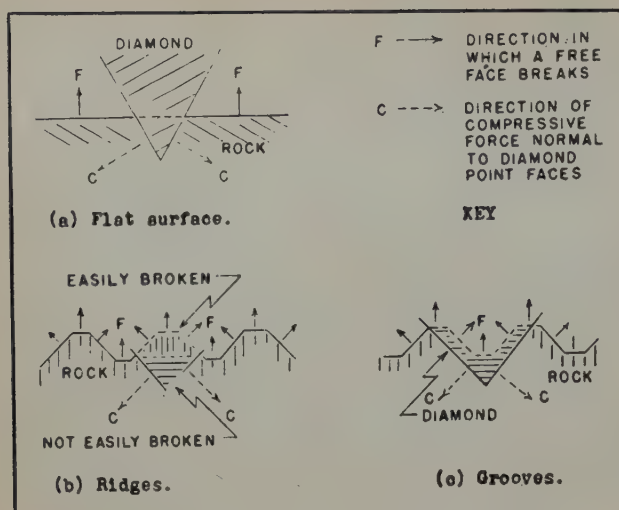


Fig. 14—Diamond point penetration of various surfaces.

limited porosity, those particles being formed by the cutting action must break toward free faces, a *free face* being defined as an exposed surface. This principle of breaking the rock to free faces, illustrated in Fig. 14, seems important in diamond drilling.

During drilling of a flat surface, only one free face is exposed, and the mineral will break with difficulty. Such a flat surface may be formed by well-worn diamonds. When diamonds are set so as to form alternate flat-top ridges and grooves, three free faces are exposed. This should permit the diamonds to break out larger rock particles and thereby increase drilling efficiency, particularly if the bit design permits ready removal of these coarser cuttings. A setting of diamonds in wide concentric grooves, wherein all the stones track in grooves cut by the preceding diamonds, would seem to be a less effective practice because of the difficulty in achieving appreciable diamond penetration.

Size and Protrusion of Diamonds: It is generally recommended 1—that small diamonds, with a slight protrusion from the matrix, be used for hard dense rocks and 2—that larger diamonds, well exposed, be employed in bits designed to drill the soft rocks such as limestones, marbles, and shales. To a large degree, the poor bit performance in the soft rocks during these experiments was undoubtedly a result of using small diamonds in the bit. The tendency to

cake was accentuated by the slight clearance between the individual diamonds and between the matrix and the bottom of the hole. Larger stones would provide more clearance for cuttings to be washed away, besides forming larger-sized particles with less tendency to aggregate into cake.

The microscopic examinations of the cutting surfaces at the bottom of the drill holes showed that the hard minerals fissure and fracture below the depth of diamond penetration. It would seem, therefore, that good drilling efficiency can be obtained by using small diamonds for hard rocks, particularly if and when the speed of rotation is increased appreciably. Furthermore, the reduced protrusion of the smaller stones will decrease diamond breakage when drilling hard rocks and minerals.

Effect of Waterways: It was evident that the Truco bits would not have become plugged with cake so easily when drilling limestone if they had had more waterways. This conclusion is based on the fact that plugging originally commenced at a place on the bit nose just before the cuttings reached a waterway. In other words, a concentration of cuttings from nearly one-half the circumference of the bit built up a cake at this critical point. It is reasonable to believe that intermediate waterways would have eliminated this concentration, provided sufficient water was employed to maintain a good flow across the bit face.

In any event, for a given optimum water volume, the size and number of waterways must be such as to permit a good flow of water to pass between the rock and bit matrix while seeking paths between the diamonds. The purpose of this water is to carry the cuttings away immediately, in their coarsest state, as well as to cool the bit. Those cuttings that are not flushed away directly by the water passing across the bit face remain to be ground finer, until they are eventually overtaken by a waterway.

Testing of Bit Design

The authors feel that such a laboratory drill unit as described in this report can be employed very effectively in the development and testing of new bit designs for various rock types. The unit has several advantages over field testing procedures, namely:

1—Results may be obtained more quickly and easily, and with better control.

2—A microscopic study of the bottom of a hole in the small test block will aid in the interpretation of

Table III. Screen Analyses of Diamond Drill Cuttings

Rock Type	RPM	Bit Force, Lb	Advance, In. Per Min	Cutting Saved, Pct	Size Range, Wt Pct				
					+ 65 Mesh	- 65 + 100	- 100 + 200	- 200 + 270	- 270
Truco Bits									
Granite	80	100	0.1	32	1	6	15	8	70
Granite	305	100	0.2	42	Trace	6	10	6	78
Granite	80	500	0.75	62	2	7	17	9	65
Granite	305	500	2.45	57	1	6	16	8	69
Trap rock	305	550	2.1	61	2	7	16	7	68
Taconite	305	700	2.2		1	17	20	8	54
Marble	305	400	2.25	57	4	6	10	4	76
Limestone	305	150	6.1	45	3	7	22	8	62
Sandstone	305	350	8.75	60	1	10	26	12	51
Ex Bit									
Granite	800	Pressure	0.8	85	1	2	10	7	80
Granite	1000	Control	1.2	82	1	2	11	5	81
Granite	1200	Inaccurate	0.6	58	1	1	7	6	85
Granite	1400		0.6	72	1	1	6	7	85

the cutting action of the test bit, suggesting corrective measures.

3—The small size trail bits should be less costly than experimental EX bits.

4—Drill cuttings may be collected very easily and used for sizing analysis and microscopic examination. Thus small-scale bits can be used to test radical designs and to suggest both minor and major changes. When a satisfactory model has been developed, a full-scale bit of this design could be tested in the field.

Summary

A study of bit performance, as well as the microscopic examination of bottom-hole conditions and the observation of the bit cuttings for the test conditions cited, led to the following interpretations as to the cutting action of the diamond drill bit:

1—Bit advance is approximately proportional to increases in rpm and/or force on bit during drilling at the slower rates. The efficiency drops off in the higher speed and pressure ranges as particles become more finely ground or the bit becomes partially plugged with a mineral which forms a cake.

2—The tendency to cake is common to the finer grained rocks, marble, limestone, and trap rock, containing such soft minerals as calcite and biotite.

3—Each mineral constituent of a rock is cut in a definite manner, depending upon its hardness, toughness, cleavage, and other physical properties. The total effect of this cutting action is modified in heterogeneous rocks by variations in mineral composition, texture, porosity, and cementation.

4—A rough correlation exists between rate of advance and size range of cuttings, i.e., the higher the rate of advance, the coarser the particle size.

5—The force on the bit causes diamond penetra-

tion by reason of compressive force; *rotary bit speed* causes an impact and a shearing force. These actions develop tension and/or shear fractures.

6—The principle of setting the diamonds to obtain drilling action that will break minerals to as many free faces as possible seems important.

7—Small diamonds, with slight protrusion, should be used for hard dense rocks; larger diamonds, well exposed and with ample water passages, should be employed in bits designed to drill soft rocks.

Acknowledgment

The authors wish to thank the officials of Region V, and the Mount Weather Station, Region VIII, of the U. S. Bureau of Mines, for their interest in promoting and aiding this study. Dr. G. M. Schwartz, Professor of Geology, and Professor T. L. Joseph, Assistant Dean, School of Mines and Metallurgy, University of Minnesota, were most helpful in reading the manuscript and in making suggestions.

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Adsorption of Calcium and Sodium on Pyrite

by A. M. Gaudin and W. D. Charles

IN flotation lime is used to depress pyrite. For this purpose it is preferred to caustic soda. The low cost of lime and the widespread availability largely account for this preference. However, there is some evidence¹ that pyrite is more readily depressed by calcium than by sodium ions at the same pH. To measure the difference in adsorption of calcium and sodium, which is thought to lie at the basis of the differences in depressing action, radionuclides of

calcium and sodium have been used. The experimental work has included an evaluation of the effect of dissolved oxygen, of the concentration of calcium or sodium, of the concentration of hydrogen ion, and of the anions present.

Preliminary work in the investigation entailed preparation of the pyrite, development of a technique for counting pyrite coated with Ca^{45} and Na^{22} , the radionuclides used, and development of an apparatus to study the adsorption of calcium and sodium on pyrite.

The main experimental work included calcium adsorption and sodium adsorption studies detailed below.

The following are the most salient observations made during these experiments. Oxygen in solution increases the adsorptive power of pyrite for calcium

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Discussion on this paper, TP 3491B, may be sent (2 copies) to AIME before April 30, 1953. Manuscript, Nov. 5, 1952. Los Angeles Meeting, February 1953.

and for sodium. The effect is greater for calcium than for sodium. The adsorption density of calcium increases with increase in calcium concentration and with pH. Sodium adsorption density increases with pH and decreases with increasing concentrations of calcium and cyanide ions. Calcium adsorption decreases slightly in the presence of sodium and/or cyanide ions. In a solution containing potassium ethyl xanthate more calcium and/or more sodium is adsorbed than in the absence of the collector.

Reagents

Conductivity water was used throughout in the preparation of solutions. It was prepared by re-distilling the laboratory supply of distilled water under an atmosphere of purified tank nitrogen. Conductance measurements of the water indicated that it contained less than 0.01 part per million of dissolved salts (expressed in terms of sodium chloride).

Table 1. Relation of Calcium Adsorption to Oxygen

Calcium Concentration, Millimols Per Liter	pH		Calcium Adsorption, Micromols Per Sq M	Solutions Exposed To:
	Influent	Effluent		
0.101	9.50	9.49	2.12	Nitrogen
0.101	9.52	7.65	3.93	Atmosphere
0.101	9.50	7.30	4.34	Oxygen
0.145	10.26	10.19	2.39	Nitrogen
0.145	10.30	9.85	7.45	Atmosphere
0.145	10.28	9.05	9.27	Oxygen

Radiocalcium: Radiocalcium was received from the National Laboratory at Oak Ridge, Tenn., as 1½ ml of a chloride solution containing 2 millicuries (mc) of Ca⁴⁵. This was diluted to 100 ml and stored in a tightly sealed Saftepak bottle. Two to four milliliters of the active solution per adsorption test was found to give the pyrite sample a sufficiently high activity for easy counting in a Q-gas counter. The calcium concentration of the diluted active solution was 8.0 mg per liter.

Radiosodium: The original sample of sodium 22, purchased from the National Laboratory at Oak Ridge, was 1 ml of active solution containing 1 mc as sodium nitrate. The milliliter of active solution was diluted to 1000 ml and stored in a tightly sealed Saftepak bottle. Ten milliliters of the resulting solution were found to give the pyrite sample a sufficiently high activity for easy measurement with a Q-gas counter. The sodium concentration of the active solution was 1.2 mg per liter.

Reagent Gases: Nitrogen and oxygen were received in steel cylinders at 1600-lb pressure and had a listed purity of 99.9 pct and 99.6 pct, respectively. The nitrogen was further purified by passing it over copper gauze at 500°C and finally through a calcium hydroxide bath.

Potassium Ethyl Xanthate: A commercial sample of potassium ethyl xanthate was purified according to Foster.⁵ The recrystallized salt was stored in an evacuated pyrex bottle and kept in a refrigerator.

Other Chemicals: Analytical reagent-grade chemicals were used in the preparation of standard solutions of calcium hydroxide, calcium sulphate, sodium hydroxide, sodium cyanide, and hydrochloric acid.

Pyrite

Coarse pyrite handsorted from large crystalline masses was crushed to 28 mesh, then ground in a

steel ball mill until all passed through 200 mesh. The ground product was deslimed by sedimentation to eliminate -30 micron particles. The 30/74 micron pyrite was washed several times with dilute hydrochloric acid, then with distilled water, and further deslimed. The cycle was repeated several times and the final product stored in a pyrex flask under dilute hydrochloric acid, pH about 2.5.

A sample of the cleaned pyrite was searched spectrographically for impurities and the following elements were found: Al, Ca, Cu, Na, to the extent of 0.01 pct or less, Ba, K, Si, to the extent of 0.1 to 0.001 pct each.

The krypton gas adsorption apparatus^{2,3} was used to determine the specific surface of the sized pyrite. The average of two measurements gave 612 sq cm per gram. With a surface factor⁴ of 2.58 and an average particle size of 52 microns, the calculated specific surface is 590 cm² per gram. This is in good agreement with the measured surface area.

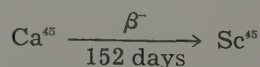
An attempt was made to synthesize pyrite containing radioactive isotopes of iron and sulphur, 1—by hydrothermal crystal-growth processes and 2—by irradiating a sample of natural pyrite in the Brookhaven pile for a period of two months. It was anticipated to trace the activities produced by the radioactive constituents in adsorption experiments and thereby help clarify the adsorption mechanism of calcium and sodium on pyrite. The pyrite crystals produced hydrothermally were extremely fine and the yield low, so experimentation along that line was abandoned. The irradiating procedure gave evidence of formation of isotopes other than radioiron and radiosulphur; therefore that method was abandoned also.

Experimental Adsorption Tests

A solution containing reagents such as calcium or sodium hydroxide, hydrochloric acid, calcium sulphate or sodium sulphate, depending on the nature of the test, was passed through a bed of clean, ground pyrite retained on a Buechner fritted-glass funnel. The mineral was then separated from the solution and the amount of reagent on the pyrite surface determined by radioanalysis. All test solutions were exposed to purified nitrogen unless stated otherwise. Details may be found in the thesis by William D. Charles.⁶

Preparation of Ca⁴⁵ and Na²² Standards: The relationship between concentration of radionuclide and activity was ascertained by counting several 5-g samples of pyrite onto which was evaporated a measured volume of active solution. The results indicated that the activity of the pyrite sample varied directly with the volume of active solution evaporated on the mineral.

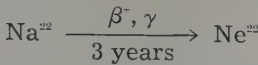
Method of Beta Counting for Adsorbed Calcium and Sodium: Calcium 45 has a half-life of 152 days and emits only β⁻ radiation. The energy of the beta particle is 0.25 Mev.,^{7,8} the decay scheme being



This radiation is sufficiently strong to be counted by an end-window type of Geiger-Muller tube, but the counting efficiency is low. Accordingly an internal gas counter was chosen, primarily for its higher efficiency. The counter chamber was operated at atmospheric pressure in a mixture of helium and butane, or Q-gas. The gas was allowed to flow slowly

through the counter to prevent the air from leaking into it.

The same instrument was used to evaluate sodium 22. This isotope has a convenient half-life of 3 years, or long enough to require no allowance for decay during the time taken by the experiment. On decay sodium 22 emits 0.575 Mev. beta radiation and 1.30 Mev. gamma radiation.^{7, 8}



Five grams of pyrite, or considerably more than enough for infinite thickness, for both beta radiations, was used throughout. Details of evaluation of total cation concentration on the surface of the pyrite may be found in Appendix I to the thesis by William D. Charles.

The activity on a sample of pyrite from the adsorption test was due in part to the cation adsorbed and in part to the cation in the solution retained by the pores of the cake of mineral, before drying. The

Table II. Effect of Sodium on Calcium Adsorption

Calcium Concentration, Millimols Per Liter	pH of Effluent	Sodium Concentration, Millimols Per Liter	Calcium Adsorption, Micro-mol Per M ²
.10	9.5	0	2.12
.10	9.5	1	2.11
.10	9.5	10	1.60

difference between the total cation determined from activity measurements and the cation in the retained solution gives the amount of cation adsorbed by the pyrite sample. The moisture retained was assumed to have had the same marked cation concentration as the bulk solution. Knowing the specific surface area of the pyrite the result was expressed as micromols of cation adsorbed per square meter of pyrite surface.

Adsorption of Calcium on Pyrite

1—Effect of Oxygen in Solution: Preliminary experiments indicated that equilibrium between adsorbed calcium and calcium in solution was not attained when the solutions and the mineral were exposed to the atmosphere. Adsorption tests at concentrations of 0.101 and 0.145 millimols calcium per liter were conducted in which the oxygen content of the solution was varied by exposing the solutions 1—to purified nitrogen, 2—to the atmosphere, and 3—to oxygen.

The data shown in Table I indicate that the amount of calcium which is adsorbed increases with the amount of oxygen in the solution. A change in pH was noted, in the direction of acid formation, and found to be greatest when the solutions were exposed to oxygen.

2—Attainment of Equilibrium: Four adsorption tests were conducted in which the only variable was the time of contact between the solution and the mineral bed. The flow rate was held constant at approximately 500 ml per hr and the tests run for 1, 2, 4, and 6 hr. The experiment was made both under conditions of exposure of solutions to the atmosphere and of exposure to purified nitrogen only.

Where contact was made with air the adsorption increased with reaction time, but with nitrogen, the adsorption was the same regardless of the volume of throughput, thus proving that under non-oxygenated

conditions a contact time of 1 hr is sufficient for equilibrium.

3—Effect of pH: Two series of tests were conducted to investigate the effect of pH on calcium adsorption. The calcium concentration in the test solutions was kept constant while the pH was varied with dilute hydrochloric acid. In one series the calcium concentration was maintained at approximately 1.45 millimols per liter while the pH of the solution was varied from 3.95 to 9.95. In the other series the calcium concentration was kept at approximately 0.145 millimol per liter while the pH varied from 3.62 to 9.80.

The results, see Fig. 1, show that the amount of calcium adsorbed increases with pH. A linear relationship between the logarithm of the amount of calcium adsorbed and the pH of the solution is suggested. For a tenfold increase in pH there is approximately a 50 pct increase in adsorbed calcium.

4—Effect of Calcium Concentration: The quantity of calcium ion adsorbed on pyrite at calcium concentrations ranging from 0.25 micromol to 1.8 millimols per liter was measured. The pH varied from 5.4 to 6.8 in spite of the attempt to maintain it at 6.0 with 0.01 N. HCl. Since the variation of calcium adsorption with pH is known, however, it is possible to adjust the calcium adsorption data to a uniform pH, 6.0. The amount of calcium adsorbed as determined in each test was corrected for deviation in pH by using the effect shown in Fig. 1. The corrected values are presented in Fig. 2.

The results indicate a consistent variation in the amount absorbed with calcium concentration in solution. In the concentration range of 0.25 micromol to 20 micromols calcium per liter, the experimental points follow a straight line with a slope of one. The deviations from the straight line appear within experimental error. Beyond a concentration of 20 micromols per liter, the surface concentration increases less rapidly and more complexly.

5—Effect of Sodium: To evaluate the effect of sodium ion on calcium adsorption, tests were made with Na⁺ at 1 and 10 millimols per liter, as well as without sodium. The sodium was introduced in the form of sodium chloride. Both the calcium ion and pH of the solution were kept constant with the

Table III. Effect of Anion Present on Adsorption of Calcium on Pyrite

Anion Present, Millimols Per Liter	Calcium Concentration, Micromols Per Liter	pH of Effluent	Calcium Adsorbed, Micromol per M ²
Blank (CaCl ₂)	144.	5.30	0.41
.144 Sulphate	144.	5.30	0.36
Blank (CaCl ₂)	1440.	5.05	1.13
1.44 Sulphate	1440.	5.05	1.20
Blank (CaCl ₂)	73.2	6.35	0.59
.013 Xanthate	73.0	6.30	1.21
.126 Xanthate	73.1	6.32	1.42
Blank (CaCl ₂)	145.	9.80	2.90
1.0 Cyanide	145.	9.80	1.97
Blank (CaCl ₂)	145.	10.30	3.60
10.0 Cyanide	145.	10.30	2.16

former at approximately 0.1 millimol per liter and the latter at 9.5. The results are summarized in Table II.

The data indicate that the sodium ion present in solution does affect the adsorption of calcium. A slight decrease in the amount of calcium absorbed is found when the sodium ion concentration is increased to 10 millimols per liter.

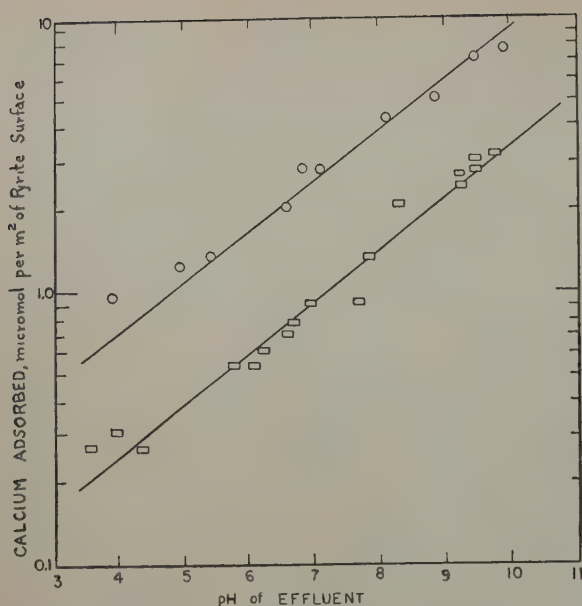


Fig. 1—Effect of pH on calcium adsorption, from solution containing 1.45 millimols of calcium per liter (represented by circles) and 0.145 millimols of calcium per liter (represented by rectangles.)

6—Effect of Variation of the Predominant Anion Present: Several tests were conducted in which the predominant anion other than chloride was varied. Standard solutions of calcium sulphate, potassium ethyl xanthate, and sodium cyanide were added and the adsorption of calcium determined. The results are summarized in Table III.

The data suggest that sulphate ion does not materially affect the adsorption of calcium on pyrite, but that calcium adsorption increases in the presence of xanthate ion and decreases in the presence of cyanide ion.

7—Reversibility of Adsorption: Three tests were conducted to verify the exchange between active and inactive calcium at the surface of pyrite. In each test 1 liter of calcium solution under a nitrogen atmosphere was allowed to pass through a bed containing approximately 7 g of pyrite. The calcium ion concentration, pH, and flow rate were held constant; only the sequence of solution addition was varied.

Table IV. Reversibility Adsorption Tests with Calcium

Solutions Added		pH of Effluent	Activity on 5 G Pyrite, CPM*	Activity of First Test on Pyrite, Pct
Solution 1	Solution 2			
Active		9.20	342.8	100
Inactive	Active	9.20	285.9	83.5
Active	Inactive	9.20	12.4	3.6

* Counts per minute.

In the first test 1 liter of active solution was allowed to pass through the mineral bed and the activity on the pyrite determined. In the second test 1 liter of inactive solution was followed by 1 liter of active solution and the activity on the pyrite determined. In the final test 1 liter of active solution was allowed to pass through the bed, followed by a liter of inactive solution, and the activity remaining on the pyrite was determined.

The data, which is summarized in Table IV, in-

dicate that almost complete exchange takes place when 1 liter of solution is passed through the mineral bed. With sufficient time of contact complete reversibility of adsorption of active and inactive calcium seems likely.

The effect of passing conductivity water through a bed of Ca^{45} -coated pyrite was also investigated. The results of two tests, in which 2 to 4 liters of water were used, are summarized in Table V. The data indicate that with a sufficient amount of conductivity water, complete removal of the adsorbed calcium would occur.

Adsorption of Sodium on Pyrite

1—Effect of Oxygen in Solution: Since it was found that oxygen in solution had a marked effect on calcium adsorption, a series of tests were conducted to evaluate the effect of oxygen, if any, on sodium adsorption.

Adsorption tests at concentrations of 1 and 0.1 millimol sodium per liter were made, in which the oxygen content of the solution was varied by exposing the solutions 1—to purified nitrogen, 2—to the atmosphere, and 3—to oxygen.

The data, see Table VI, indicate that the amount of sodium adsorbed increases with the amount of oxygen in solution; however, this effect is not nearly as great as in the case of calcium when the solution contained dissolved oxygen.

2—Attainment of Equilibrium: Four adsorption tests were conducted in which only the time of contact between the solution and the mineral bed

Table V. Leachability of Adsorbed Calcium

Ca Concentration in Activating Liquor, Micromols Per Liter	Cond. Water Used, Vol.	pH of Effluent	Activity on 5 G Pyrite CPM	Activity of First Test on Pyrite, Pct
0.6		6.1	2445	100
0.6	2 liters	6.0	90.7	3.7
0.6	4 liters	6.0	55.0	2.3

was varied. The sodium concentration, pH, and flow rate (~500 ml per hr) were held constant. The solutions were exposed to purified nitrogen and the tests run for 1.0, 1.5, 2.0, and 4.25 hr. The data indicate that a contact time of 1 hr is sufficient for equilibrium to be reached between adsorbed sodium and sodium in solution. There was substantially no change in the pH of the solution passing through the pyrite bed.

3—Effect of pH: A series of tests was conducted to investigate the effect of pH on sodium adsorption. The sodium concentration in the test solutions was kept constant at approximately 0.101 millimol per liter while the pH was varied with dilute hydrochloric acid. The results obtained are shown in Fig. 3. A linear relationship between the logarithm of the amount of sodium adsorbed and the pH of the solution is suggested. For a tenfold decrease in hydrogen ion concentration there is approximately a 40-pct increase in adsorbed sodium.

4—Effect of Calcium: To evaluate the effect of calcium ion on sodium adsorption, tests were made with calcium at 1 and 10 millimols per liter, as well as without calcium. The calcium was introduced in the form of unmarked calcium chloride. Both the

sodium ion and pH of the solution were kept constant, with the former at approximately 0.10 millimol per liter and the latter at 10.8.

The results, given in Table VII, indicate that calcium ion has a very marked effect on the adsorption of sodium by pyrite. The amount of sodium adsorbed decreases rapidly with increasing calcium concentration.

5—Effect of Variation in the Predominant Anion Present: Several tests were conducted in which the predominant anion other than chloride was varied. Gaged amounts of sodium sulphate, sodium cyanide, and potassium ethyl xanthate were added and the adsorption of sodium determined. The results obtained are summarized in Table VIII.

The data indicate that the adsorption of sodium is not materially affected by the sulphate ion present in solution. However, it was found that the adsorption density of sodium increases in the presence of

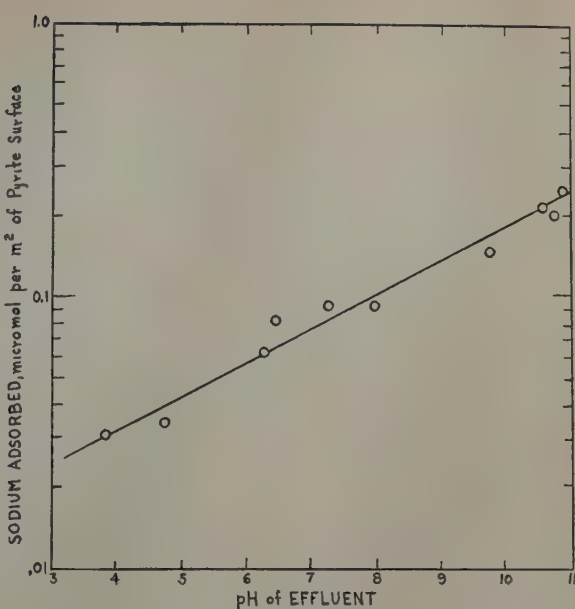


Fig. 3—Effect of pH on sodium adsorption, from a solution containing 0.101 millimols of sodium per liter.

Table VI. Effect of Oxygen on Sodium Adsorption on Pyrite

Sodium Concentration, Millimols Per Liter	pH		Sodium Adsorption, Micro-mols Per M²	Solutions Exposed to:
	Influent	Effluent		
0.10	8.05	8.02	.086	Nitrogen
0.10	8.02	7.50	.112	Atmosphere
0.10	8.03	7.20	.115	Oxygen
1.0	10.28	10.27	2.14	Nitrogen
1.0	10.28	10.25	2.34	Atmosphere
1.0	10.28	10.20	2.66	Oxygen

xanthate ion and decreases in the presence of cyanide ion. Qualitatively, the results are parallel to those obtained with calcium.

Discussion of Results

Our experiments⁶ have shown that oxygen in solution increases the adsorption of calcium and/or of sodium by pyrite. The non-equilibrium condition which exists when the solutions are open to the atmosphere seems due to the constant alteration of the mineral surface by dissolved oxygen, viz., the continuous formation of oxides of sulphur and perhaps of iron at the pyrite surface. The pH of the effluent from the adsorption column is lower than that of the influent when the solution contains dissolved oxygen but is unchanged when oxygen is excluded. This is probably because of dissolution of the oxy-acids of sulphur that form on oxidation

of the mineral. The effect of dissolved oxygen was greater on calcium adsorption than on sodium adsorption. This is in line with what is known of the solubilities of calcium sulphate and sodium sulphate. It would certainly be enlightening at this point had it been possible to compare the effect of dissolved oxygen on the adsorption of a cation having a sulphate less soluble than calcium, e.g., strontium, barium, or radium. Unfortunately, time was not available to explore this suggestion.

Since the dimensions of the pyrite crystal lattice are available,¹¹ it is possible to calculate the per cent coverage by adsorbed cations from the surface-area determination (612 cm²/g). The requirements for a complete monolayer is assumed to be one cation per pair of sulphur atoms or per iron atom. Assuming that the surface of crushed pyrite consists of an assortment of 100 planes, and on the basis of the accepted lattice, the area per site is found to be 14.5 Å². This calculation yields 11.4 micromols per

Table VII. Effect of Calcium on Sodium Adsorption on Pyrite

Sodium Concentration, Millimols Per Liter	pH of Effluent	Calcium Concentration, Millimols Per Liter	Sodium Adsorbed, Micromols Per Sq M
0.10	10.8	0	0.198
0.10	10.8	1	0.040
0.10	10.8	10	0.027

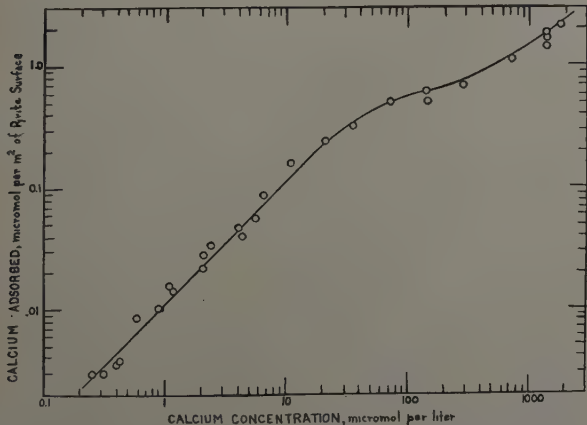


Fig. 2—Effect of calcium concentration on the adsorption of calcium on pyrite. Corrected to pH = 6.0.

sq m for a complete monolayer. Therefore the surface covered, at the highest concentration of calcium, 1.8 millimols per liter, is approximately 18 pct. At a concentration of 0.1 millimol per liter and a pH of 6.0 the comparative amounts of surface coverage for calcium and sodium on pyrite is found to be 4.5 and 0.35 pct, respectively. There was no experimental evidence, within the limits of this investigation, to indicate that a sufficient amount of cation was ever adsorbed to form a complete monolayer.

The results of surface-covered calculations are quite different, however, if the cations are assumed

to be adsorbed as hydrated ion. The cations are known to exist in aqueous solution in highly hydrated forms; therefore their adsorption as such may be the governing factor when their configuration on the pyrite surface is considered. The radii of the hydrated cations reported in the literature vary; Nachod and Wood¹² give 6.7 Å for calcium and 7.8 Å for sodium. When these figures are used, the amount of calcium or sodium required to form a complete monolayer of densely packed hydrated ions is 1.2 and 0.9 micromols per sq m, respectively. At a concentration of 0.1 millimol per liter and a pH of 6.0, the comparative amounts of surface coverage, hydrated cation basis, on pyrite are 40 pct for calcium adsorption and 4.0 pct for sodium. Also, the amount of calcium adsorbed at an equilibrium concentration of 0.64 millimol per liter is sufficient to form a complete monolayer of the hydrated ions.

The adsorption of calcium, as illustrated in Fig. 2, increases linearly with concentration up to a point where the curve begins to level off, and then appears to increase again approaching a slope of unity. Such a change in slope would seem to indicate that the character of the substratum on which calcium is adsorbed had changed. This is possibly the case, for the curve begins to deviate from the linear relationship as it approaches the point corresponding to a monolayer of densely packed hydrated calcium ions.

In the concentration range of 0.12 to 20 micromols of calcium per liter the equation of the adsorption curve, Fig. 2, may be expressed as follows:

$$\Gamma = k C$$

where Γ is the amount of calcium adsorbed per unit surface, C the equilibrium concentration of calcium in solution, and k a constant. This is the Langmuir adsorption equation where in very dilute solutions or at small surface coverage the adsorption density is directly proportional to the concentration in solution. At high concentrations the adsorption of an

Variation in the concentration of hydrogen ion affects slightly the adsorption of calcium and sodium, see Figs. 1 and 3. The experimental curves indicate a surprisingly small effect which may be expressed as follows:

$$\log \Gamma_x = \log \Gamma_7 + n (x - 7)$$

in which n is the slope of the line, Γ_7 is the adsorption at pH 7, and Γ_x the adsorption at pH = x . In the results, n is equal to about 1/6 for the calcium adsorption data and about 1/8 for the sodium data. A similar relationship exists for the variation of barium⁹ and sodium¹⁰ adsorption on quartz with variation in hydrogen ion concentration, n being equal to about 1/5 for the barium adsorption data and about 1/6 for the sodium data.

The fact that calcium and sodium ions affect one another in their adsorption on pyrite indicates competition between the cations for the same sites. Experimental evidence has shown that calcium adsorption is less affected by sodium ions than is sodium adsorption by calcium ions. The more pronounced affinity of calcium for the pyrite surface is further justified by the fact that calcium adsorption is greater than sodium adsorption at the same concentration and pH.

Acknowledgment

The authors wish to acknowledge the substantial help given them by their associates at Massachusetts Institute of Technology, as well as the fact that this work was carried out under an unclassified research project in Mineral Engineering sponsored by the U. S. Atomic Energy Commission.

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Table VIII. Effect of Anion Present on Adsorption of Sodium on Pyrite

Anion Present, Millimols Per Liter	Sodium Concentration, Millimols Per Liter	pH of Effluent	Sodium Adsorbed, Micromols per M ²
Blank (NaCl)	0.10	6.3	0.081
0.05 Sulphate	0.10	6.3	0.084
Blank (NaCl)	1.0	6.0	0.56
0.505 Sulphate	1.0	6.0	0.54
Blank (NaCl)	0.10	9.0	0.125
0.069 Xanthate	0.10	9.0	0.130
0.126 Xanthate	0.10	9.0	0.243
Blank (NaCl)	0.10	10.5	0.201
0.051 Cyanide	0.10	10.5	0.131
Blank (NaCl)	1.0	10.3	2.15
0.90 Cyanide	1.0	10.3	1.71

electrolyte cannot be expressed by such a simple relationship.¹³ The electrical effects produced at the interface by ionic adsorption then become important. It is quite possible that the deviation of the experimental curve from a slope of unity at concentrations greater than 20 micromols of calcium per liter may be rationalized in that fashion.

The Selection of Detachable Drill Bits

by E. R. Borchardt

IT is notable that the first large-scale mine operation equipped entirely with detachable bits was the Badger State mine of the Anaconda Copper Mining Co. in Butte, Montana, just 30 years ago. This mine in 1922 was producing approximately 1200 tons of ore per day. Much of the data presented in C. L. Berrien's article¹ describing the development and installation of the Hawkesworth detachable drill bit were obtained from these operations.

As in any pioneering effort, no precedent existed and many difficult problems required solution, so that the changeover to detachable bits at all Butte hill mines was not completed for 6 years. There was widespread disbelief as to the probable efficiency of the new installation.

Some attempts were made in 1931 by the owners of the Hawkesworth patents to interest Ontario gold mine operators in the bit. These efforts were not successful, but they undoubtedly stimulated thinking which resulted in the invention and patenting of several well-known Canadian detachable bits, one of which is now a widely used throwaway bit. The success of the Butte installation also led to the development of the threaded type of bit connections by several well-known manufacturers, and in 1935 these bits were introduced to the mining industry on a national scale.

The original Hawkesworth bit was not provided with a water hole but depended upon water passing through the clearance opening between the tongue in the bit and the groove in the rod to flush cuttings from the drill hole, see Fig. 1. In December 1935 it was found that this method of introducing drilling water to the bit face resulted in high dust counts. To correct this a water hole was drilled on the central axis of the bit, passing through the tongue. Unfortunately, quenching water would rise through the small water hole, spot-hardening the tongue to cause breakage, never completely eliminated.

In the fall of 1936 large-scale tests indicated that savings would be effected by use of a threaded type of bit, which was therefore adopted as standard for all Butte mines. This type of bit was used until 1947, when it was superseded by a one-use slip-on type.

Since the first use of the Hawkesworth bit every detachable bit of importance has been investigated, and where advantages which might reduce costs or increase efficiency were indicated, substantial tests

of the bit were carried on in the Butte mines. When tests demonstrated the advisability of changing from one kind of detachable bit to another the change was made at one level or in one area each day until the new rod and bit equipment was used throughout the mine. This involved a minimum of cost and disruption of drilling.

Intelligent selection of a detachable bit to obtain optimum results requires careful consideration to achieve a balance between the three principal types of equipment used in the drilling process: 1—drill bits, 2—drill steel, and 3—drilling machines. Optimum results imply maximum output and minimum cost per unit of output. Since every rock type differs in drillability and it is generally impractical to provide equipment for more than one or two types of rock which may occur in one operation, selection of equipment must encompass average drilling conditions. However, on exceptional occasions several widely differing conditions may make it mandatory to provide equipment best suited to each condition.

The choice of rock-drilling equipment is a most controversial subject and one that is further complicated by unreliable and frequently misleading performance claims. Small operators without the means for making accurate evaluations of equipment frequently suffer from these over-enthusiastic claims.

It is apparent from experience in rock drilling throughout the world that rock drillability is not alike in any two places, and that selection of proper equipment can only be made after conducting thorough trials of various types of equipment.

Some recent drilling tests in tactite and hornstone at the Darwin, California mine of the Anaconda Co. present some interesting clues on rock drillability. Microscopic examination of thin sections of these rocks reveals that mineral composition and rock texture are equally important in governing drillability. The Darwin hornstone is at times so abrasive that the carbide bit cutting edges become flattened to $3/32$ in. in 2 to 4 ft of drilling, and some carbide bits were dulled to this point after 9 to 10 in. of drilling. This wear was determined to be the proper point for resharpening to eliminate carbide insert breakage or breakage of the steel rod when drilling with $1\frac{1}{2}$ to $1\frac{3}{8}$ -in. bits, with a drifter of $2\frac{3}{4}$ -in. diam and 90 to 100 psi air pressure, see Supplement A.

Before considering the merits of various bit designs it may be well to review the mechanics of drilling rock with percussion drills. A sharp bit cuts by penetration and chipping. The amount of penetration governs the amount of chipping and depends upon the contact area of the cutting edge, the foot-

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Discussion on this paper, TP 3444A, may be sent (2 copies) to AIME before April 30, 1953. Manuscript, Jan. 18, 1952. New York Meeting, February 1952.

pound blow of the drill hammer, and the penetrability of the rock in terms of hardness and toughness. This in turn is dependent upon the texture and mineralogical composition of the rock. A simple illustration may serve to explain the relationship between contact area of the bit and the foot-pound blow for maximum penetration. A fine pointed tack will readily penetrate a piece of hard oak with one blow of a light tack hammer, but if a 60-penny spike is substituted for the tack, no matter how hard the spike is struck with the tack hammer it will not penetrate the oak. However, if a single jack is substituted for the tack hammer, a sharp blow will cause the 60-penny spike to penetrate the oak readily. So it is with a bit that the area or length of the cutting edge is one of the factors determining degree of penetration in a given rock, and so it is that the larger the area or length of cutting edge the greater the resistance to penetration and the larger the foot-pound blow or the heavier the rock drill hammer required for penetration in a given rock. Fig. 2 illustrates the sequence of penetration and chipping performed by the cutting edge of a bit. As the cutting edges advance through a given arc in succeeding hammer blows of the drill, the material between the indentations *a* and *c* forming the segment of a circle is detached from the rock mass by shearing or chipping. The size of segment chipped is governed by the lead of the rifle bar and is entirely dependent upon the brittleness or toughness of the rock and its hardness or penetrability. In tough rock it may not be possible to chip the segment *a-c*, in which event it would be necessary to reduce the rotation lead to point *b* to accomplish chipping of a smaller segment, *a-b*. The larger the segment chipped the larger the resultant cuttings will be, within clearance limits of the bit, and the faster and more efficient the drilling process becomes. Resistance to penetration can be altered by lessening the angle of the cutting edges or by decreasing the contact area of the total length of the cutting edge or edges. In a rose bit this length will obviously be maximum, and in a single chisel bit it will be minimum for a given gage. However, the total cutting edge length of a standard cross bit may be reduced by removal of the center portion of the cutting edge which normally does little work. To illustrate this, see Fig. 2, a plan view of a 1½-in. drill hole is divided into three circle areas, the radii of which increase by ¼-in. intervals. The area of the central circle is 0.19635 sq in. The No. 2 area is 0.58905 sq in., and the No. 3 or outer circle 0.98175 sq in. This shows that the middle or No. 2 area requires three times as much work on the part of the bit as the central

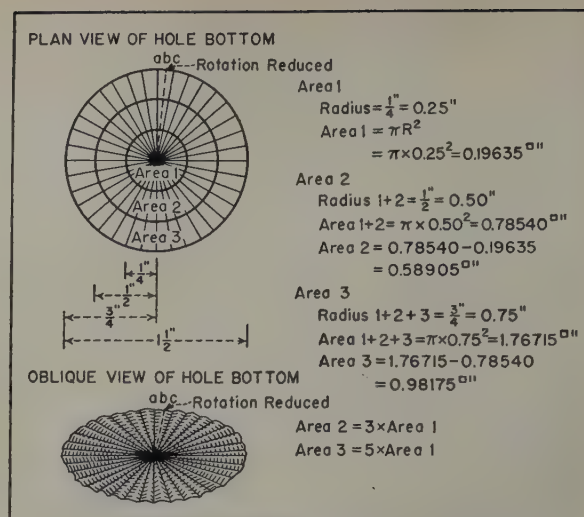


Fig. 2—Theoretical drilling impression with crossbit.

area, No. 1, and that the outside area, No. 3, requires five times the work that is required of the bit to excavate the central portion. Thus it is obvious that the ends of the cutting edges take the brunt of the wear on the bit, and in abrasive types of rock this wear may be supplemented by additional abrasion of the gage surfaces from rotation of the bit.

Usually a bit must be discarded when its cutting edges are flattened by abrasion to the point where penetration and resultant chipping cease and bit penetration is entirely by crushing, or, when the gage surface becomes worn to a reverse taper and the bit binds in the hole, see Fig. 3. The proper discard point will vary with the physical characteristics of the rock drilled. During the cutting and chipping period with sharp cutting edges the cuttings are coarse and the energy transmitted to the drill bit by the drill hammer is absorbed in penetrating and cutting the rock. However, when the cutting edges become excessively flattened, penetration is decreased, the cuttings become very fine, and a great deal of the energy of the drill hammer is dissipated in heat, which in tough drilling rocks notably increases the temperature of the drilling water issuing from the hole. At this point the advance of the bit is largely by crushing. The low penetration resulting from blunted cutting edges sets up excessive vibration in the steel rod, causing early breakage of the drill rod. Whether drilling is done with a carbide insert rod, a carbide insert detachable bit, or a detachable steel bit, continued drilling with excessively flat cutting edges in hard, tough drilling rock will result in one of four types of failure: 1—breakage of carbide insert, 2—breakage of the detachable bit skirt, 3—failure of the connection, or 4—breakage of the drill rod, see Figs. 4-9.

There are of course types of rock in which the cutting edges are scoured or hollow-ground and become sharper with continued drilling. Other types of rock yield no over-drilling, and gage wear and binding occur before the cutting edge is dulled, see Fig. 10.

Dulling of the bit is progressive, and is evidenced by reduction in drilling speed. Whether this reduction is abrupt or gradual depends upon the rock drilled and the bit design. The fact that the average drilling speed of a carbide bit is much higher than that of a steel bit in many types of difficult drilling



Fig. 1—Hawksworth bit and shank.



Fig. 3—Dulled bit showing reverse taper and flattened cutting edges.

ground is attributable to the greater resistance of the carbide bit to dulling. The steel bit will drill exactly as fast, in some cases faster, in the first few seconds of use, but its speed drops off rapidly as wear of the cutting and gage surfaces takes place. In medium or soft ground this advantage in increased drilling speed over the steel bit is not apparent. The high resistance to dulling shown by carbide inserts results from the inherent high wear resistance of the cemented carbide, which possesses a combination of high scratch resistance and high indenting resistance. Hardness of commercial carbides are in the range of 85 to 93 Rockwell A Scale, whereas steel bits range in hardness from 62 to 66 Rockwell C Scale. A reading of 65 C Scale is equivalent to a reading of 83.9 A Scale.

With few exceptions, basic bit designs have changed very little, and the relative merits of various bit designs have been well evaluated in various types of rocks during the past 40 years. A standard cross bit with and without variations is by far the most popular bit used in the world today. Variations consist of wide or narrow wings, cutting edge angles from 90 to 120°, large center hole, undercut wings, exaggerated clearance grooves for cuttings, and special

adaptations embodying pilot cutting faces. The proper cutting edge angle depends upon the friability of the rock drilled. Too large an angle results in lowered penetration and hence sets up excessive rebound of the steel. Large cutting edge angles increase the gage surface, desirable in abrasive and friable rock. Large gage surfaces are obtained by widening the bit wing in addition to increasing the cutting edge angle. Where maximum penetration is desirable in very friable rocks and gage wear is minimum, a narrow wing is advantageous.

Even the position of the water hole influences drilling speed, as in some types of hard ground a change from side to center hole resulted in a decided increase in drilling speed. In types of ground where plugging of bits and rods is a problem, a side hole bit is helpful, see Fig. 4 and Table I.

In some types of rock it is desirable to use an undercut wing in which only the upper part of the gage surface is fully hardened; the remainder, being softer, gives strength and backing to the wing, and being soft wears at the same rate as the smaller hardened surface. Thus the bit loses gage without the usual reverse taper and consequent binding.

The cross bit has been applied successfully in nearly every type of rock. But if provision for clearance and removal of cuttings is insufficient, especially in small detachable bits, and in very friable rocks that are rapidly cut, a chisel bit may be superior because it has the advantage of excellent cuttings clearance. This feature is particularly valuable in drilling down holes. The present popularity of the chisel bit has been enhanced by its use with

Table I. Results of Bit Tests in Clay-Rich Butte Rock

Center Hole Bits	Statistics	Center and Side Hole Bits	Statistics	Side Hole Bits	Statistics
Footage drilled	446	Footage drilled	1526	Footage drilled	664
No. of steel plugged	16	No. of steel plugged	30	No. of steel plugged	6
Footage drilled per plug	27.9	Footage drilled per plug	50.9	Footage drilled per plug	110.7

Swedish insert rods. The adoption of this type of bit was perhaps influenced by the fact that it could be readily and accurately resharpened and the fact that a single cutting edge requires a minimum of car-

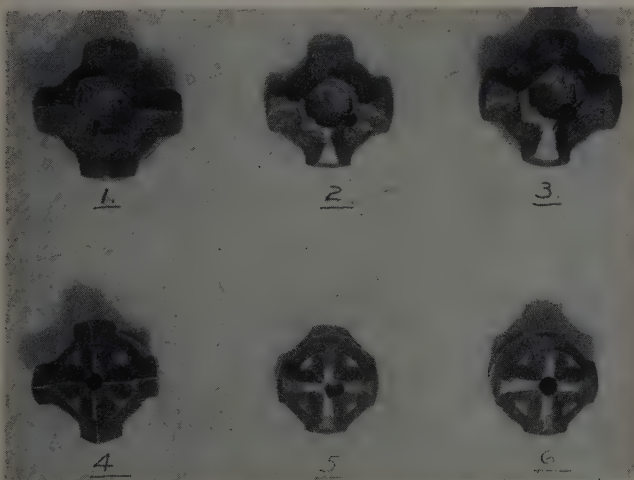


Fig. 4—(1) Sharp 2L Liddicoat bit. (2) Dulled 2L Liddicoat bit. (3) Excessively dulled 2L Liddicoat bit. (4) Sharp 1D Liddicoat bit. (5) Dulled 1D Liddicoat bit. (6) Dulled 1D Liddicoat bit.

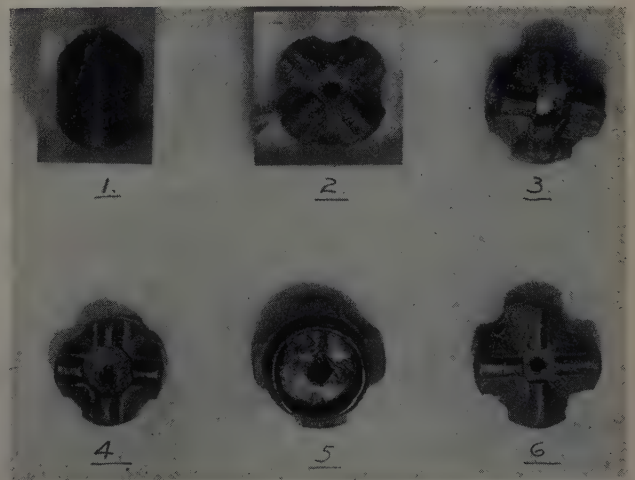


Fig. 5—(1) Coromant insert broken by excessive dulling. (2) Excessively dulled Timken bit with broken skirt. (3) Excessively dulled Liddicoat carbide bit, insert failure. (4) Excessively dulled Liddicoat carbide bit, broken skirt. (5) Excessively dulled Liddicoat 2L bit, rod tip broken. (6) Timken bit at permissible dullness.

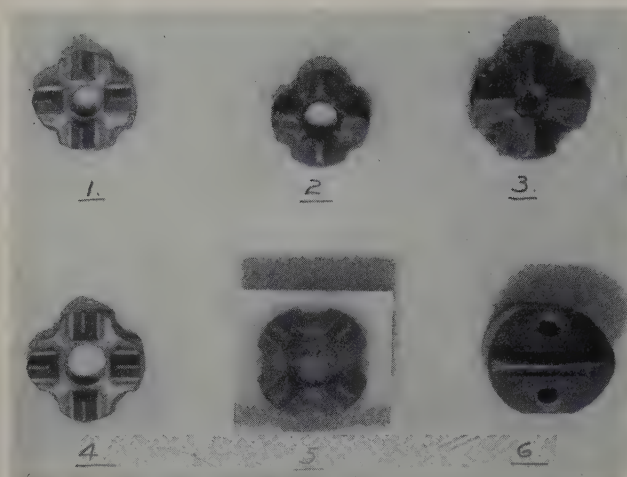


Fig. 6—(1) New Liddicoat bit face, experimental. (2) Same bit at permissible dullness. (3) Timken bit at permissible dullness. (4) New Liddicoat bit face, experimental. (5) Same bit at permissible dullness. (6) Kennametal bit at permissible dullness.

bide. The chisel bit is well adapted to uniform ground but in vuggy and broken ground it is not so efficient as the cross bit. Experiments are now under way which give promise of substantial increases in drilling speed with a re-designed chisel bit.

Both chisel and cross-type bits are available in detachable steel bits, detachable carbide insert bits, and carbide insert steel. Threaded-type detachable multi-use steel bits and carbide insert bits are supplied by two American manufacturers. The exact and uniform hardening of single-use bits is undoubtedly the major reason for the increasing popularity of the slip-on bit. With these bits it is almost unheard of to encounter hardening failures characteristic of soft bits, whereas with multi-use types, soft bits were frequent even where large-scale electrically heated salt pots were provided for heating. Apparently operators have become unwilling to place responsibility for proper hardening of bits in the hands of temperamental and frequently itinerant blacksmiths.

Detachable steel bits of the single-use slip-on type

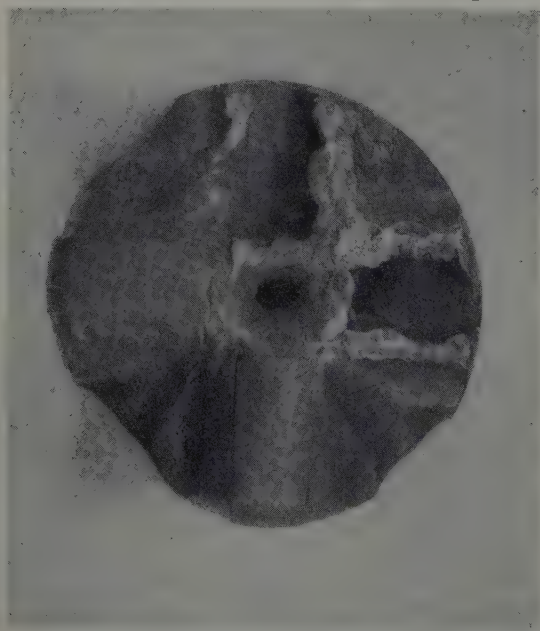


Fig. 8—Ingersoll Rand Carset bit excessively dulled, resulting in braze failure.

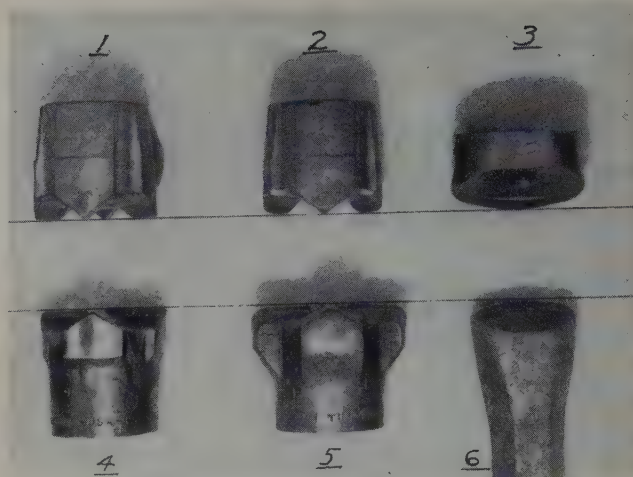


Fig. 7—(1) New Liddicoat bit profile, experimental. (2) Same bit in profile showing reduction of carbide. (3) Kennametal chisel bit showing crown and reduction of carbide on outer edges at permissible dullness. (4) New Timken bit in profile. (5) Similar bit in profile showing reduction in carbide at permissible dullness. (6) Dulled Coromant insert.

are available from several manufacturers, one of whom will shortly place on the market a slip-on type of carbide insert bit.

Insert steel is obtainable from several American manufacturers and one Swedish importer. The choice between carbide insert steel and carbide detachable bits is one that should be determined by actual test of each type to evaluate the advantages and disadvantages of each. Damage to the carbide of an insert steel results in the loss of a \$19 to \$20 rod, whereas damage to the carbide in a detachable bit results in loss of the bit only, amounting to \$7 to \$14 for small bits. Also the matter of transportation for resharpener a rod or a single bit is a subject for investigation. The manufacturers of insert steel claim the advantage of smaller bit sizes without reduced clearance, whereas the threaded insert bit requires additional skirt thickness which detracts from its clearance. It is claimed also that the life of a threaded connection is shorter than that of the carbide inserts or the steel rod. Recent tests indicate that a new slip-on type of carbide insert bit used with alloy steel rods overcomes both these objections.

As to the relative merits of carbide insert bits and steel bits, it is widely agreed that in hard, tough abrasive rock, carbide generally has a distinct economic advantage. However, there are exceptions to the rule. In some extremely hard and friable rocks it has been found impossible to drill with carbide insert bits at a reasonable cost, yet these can be drilled with single use steel bits at reasonable cost.² And there are no doubt exceptional circumstances in which carbide would have advantages over steel bits in soft ground.

A rule gaining popularity in Canada, to which of course there are exceptions, is that carbide bits are not necessary where drilling speeds of 10 in. per min and over are obtainable with steel bits.

Results of large-scale tests of carbide insert detachable bits and carbide insert steel conducted at the Sullivan mine of the Consolidated Mining and Smelting Co. of Canada, Ltd. were presented at the Annual Western Meeting of the Canadian Institute of Mining and Metallurgy, November 1950, by J. W. Reynolds. These results are believed to be of sufficient interest to be quoted verbatim.

"As a result of controlled and operational tests,

plus the experience gained from the use of tungsten carbide for all chert drilling for a period of over two years, certain conclusions may be drawn.

"1—Performance with carbide bits exceeds that with steel bits in hard ground and is about the same in medium ground and in soft ground so far as penetration speeds and rate of advance are concerned.

"2—The cost of drilling with carbide bits is definitely lower than with steel bits in hard ground, is about the same in medium ground, and is higher in soft ground. The higher cost in soft ground is mainly attributable to the fact that by comparison with their respective bits the carbide bit connection is greatly inferior to the carbide bit while the steel bit connection is superior to the steel bit. As a result so much of the potential of the carbide bit is wasted in soft ground through discard, before the carbide is finished, that the steel bit is more economical. Tungsten carbide has not yet been supplied in the universally accepted bit for use in all rock types.

"3—The undesirability of supplying large quantities of tipped rods has hindered adoption of inserted rods in place of bits in hard ground, especially since attachment failure is such a negligible factor there. The use of tipped rods for drilling in soft ground may be advantageous and is being investigated.

"4—Achieving the best performance from tungsten carbide bits has required changes in rock drills to those with fast rotation and pneumatic feed.

"5—Obtaining the best performance from carbide bits requires best obtainable drill rod performance. Alloy steels are required but their use involves rigid standards of shop practice.

"6—Successful employment of carbide bits is dependent greatly on detailed education, strict supervision, and rigid control both with distribution and use."

At other Canadian mines carbide bits improved drilling operations, but at a cost greater than was compensated for by the reduction in man hours of labor.³

The inherent strength of cemented tungsten carbide is limited; it has been proved to have sufficient strength for percussion drilling within limits, but certain measures must be taken to obtain satisfactory results. One mine lists the following precautions to be observed with the use of threaded carbide insert detachable bits.⁴

1—Insistence on the tightening of bits on rods by Stilson wrenches after each use.



Fig. 9—Tungsten carbide cutting edge at maximum permissible dullness. X30.



Fig. 10—Dulled bit showing reverse taper and sharpened cutting edges.

2—Instruction that the machine should not be turned on until the bit is tight against the bottom of the hole when changing to longer lengths.

3—Orders that bits be returned for reconditioning when 3/32-in. wear shows on the cutting edge.

4—The use of .005-in. undersize ring gage which is issued to the miner to control rod discard.

5—The use of ring gages by shift bosses to double-check the miners' rods at least once a day.

It has been found that drill steel life is governed directly by bit size; the larger the bit the less footage drilled before steel fails from fatigue, see Fig. 11. It has long been known that drilling speed increases inversely as the square of the diameter of the hole. For both these reasons it is desirable to use a small starting bit which will result in the minimum-size hole bottom consistent with satisfactory breaking. In hard abrasive rock the use of a steel bit requires larger starter sizes than with carbide bits and a greater number of steel changes, whereas the hole can be bottomed with one or two very small gage change carbide bits so that for all practical purposes the hole is the same diameter from top to bottom. The fact that it can drill a small hole quickly in hard abrasive rock is one of the major advantages of the carbide bit. The decision to use carbide or steel bits is governed entirely by cost comparisons derived from careful tests in all the various drilling conditions encountered.

Drill steel sizes vary from 7/8-in. hexagon and quarter-octagon to 1 1/2-in. hollow round, from 7/8-in. to 1 1/4-in. steel ordinarily being used in underground mining operations. In general the larger the bit size the larger the steel section required, and the larger the steel section the heavier the individual rod. Where underground transportation is involved, weight is an important consideration. Also, the heavier drill steel requires a larger foot-pound blow and a heavier rock drill. Consideration must also be given to the fact that the larger the section the smaller the number of rods which can be made up from a ton of steel, see Fig. 12.

Until the development of satisfactory alloy hollow drill steel, use of the 7/8-in. section was not entirely satisfactory. Carbon steel in this size lacked stiffness and became excessively bent and kinked from rough handling underground. Furthermore, its limberness in long lengths caused it to absorb sufficient energy

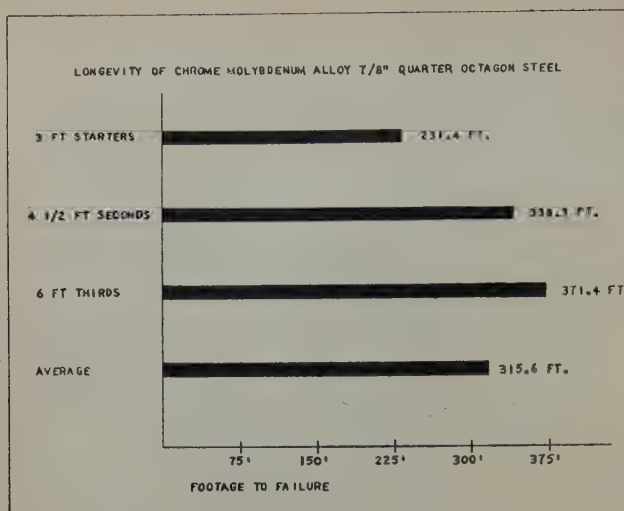


Fig. 11—Rod breakage, a factor of bit size.

of the hammer blow to result in a substantial drop in drilling speed. This condition was corrected experimentally by *skin hardening*. Alloy steel rods possess greater stiffness and have eliminated this trouble, so that the $\frac{7}{8}$ -in. quarter-octagon section with slip-on bits makes small-hole drilling feasible.

Employment of round and hexagon steel sections requires that lugs or collars be forged, and this forging and subsequent heat treatment set up strains in the rods which result in ultimate failure. In Africa, for this reason, rubber and steel collars are used. Quarter-octagon steel is generally used in anvil block-type drills requiring straight shanks with no collars or lugs.

Again, choice of steel section is dependent upon rock conditions. The use of shankless quarter-octagon rods is general with stopper drills but its use in drifters might be impractical in soft, vuggy, or ravelly ground in which bits and steel are removed from the hole with some difficulty. However, if miners become accustomed to its use most types of rock can be drilled satisfactorily with drifters using quarter-octagon steel.

In stopper drilling employment of $\frac{7}{8}$ -in. quarter-octagon steel is particularly advantageous as compared to 1-in. quarter-octagon, since because of its reduced weight it is possible for a miner changing steel to hold the drill in one hand and grasp and place a long steel in the bottom of the hole with the other hand. The weight of long 1-in. quarter-octagon rods does not permit this practice, so it becomes necessary to stand the drill against timber or a wall or lay it down while long steel changes are made.

A new development opening up an entirely new range of possibilities in drilling practice is the use of high carbon chrome molybdenum alloy hollow drill steel in the as-rolled condition, that is, in the unhardened condition as received in bar lengths from bar stock. The rods are cut into proper lengths from bar stock with abrasive cutoff wheels. Chuck ends are then chamfered on a grinding wheel and the slip-on connections are formed with a special grinder. This practice eliminates strains set up in the steel by heat treating, thus assuring maximum steel life. The consequent elimination of heat-treating equipment results in greatly reduced rod-preparation costs. Exhaustive tests of these rods with steel bits showed an increase of about five to six times the rod life, see Fig. 13, of straight carbon hollow drill steel. The use of these rods with carbide

insert bits of the slip-on type should prove a most interesting combination, especially in view of the fact that a $1\frac{1}{4}$ -in. diam bit may be used, which is somewhat smaller than usual in bottoming of holes used for breaking, except in boulder plugging. Some mines bottom holes at $1\frac{3}{8}$ -in. diam, but average American practice seems to be between $1\frac{1}{2}$ and $1\frac{5}{8}$ in.

The third factor influencing bit selection is the drilling machine, which again is determined by average rock conditions. In tough abrasive rock the strong rotation of a large bore drill to permit making a full gage change or the necessity for the great penetrating power of a heavy drill may be a determining factor. Where extreme variations in rock conditions exist it is usual to provide drills best suited to each type of rock. Experiments with drills of various bores and perhaps various rotation speeds are the only satisfactory means of determining the most efficient rock drill, bit type, and drill steel combination. In general, where rock is abrasive and gage wear is a problem, a slow rotation is helpful, but in rock which overdrills and is not particularly abrasive a fast rotation may increase the drilling speed materially. It should be pointed out that the maintenance costs of heavy drills are higher than those of light drills; therefore it is not good practice to select drills larger than required. The need is for sufficient drilling speed to permit drilling the holes required to break a round for a maximum cycle. For example, it would be uneconomical to choose a heavy fast drill with which 10-ft rounds can readily be drilled and broken, when the breaking of an 8-ft round supplies the maximum amount of broken material that can be handled on cycle. The increased weight of heavy equipment is also a factor for consideration. The trend is towards small bits, both steel and carbide, and lighter drills. However, in tunnel contracts where extreme speed of advance is essential and the use of jibs and jumbos makes weight less important, the opposite is true and the trend is perhaps towards heavier drills with larger bits.

Because of the limited inherent strength of cemented tungsten carbide, especially when it is used with small gage bits, where unit impact stresses are high, the foot-pound blow must be kept within safe limits to prevent undue fracturing of the carbide, and smaller bore, lighter drills are indicated.

In extremely difficult drilling conditions where small carbide bits are necessary a $2\frac{1}{2}$ -in. to $2\frac{3}{4}$ -in. drill is very satisfactory. Larger drills may be used with larger bits having larger and heavier carbide

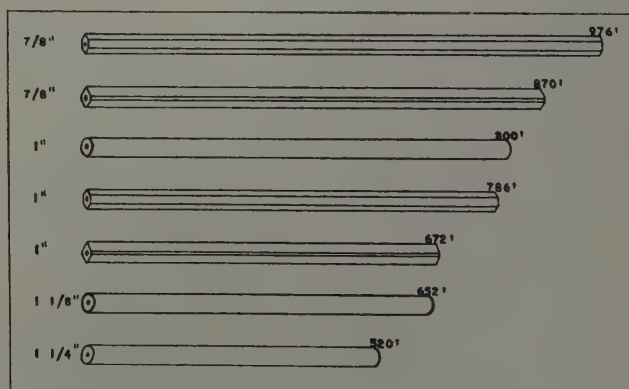


Fig. 12—Footage of drill rods that can be fabricated from a ton of steel.

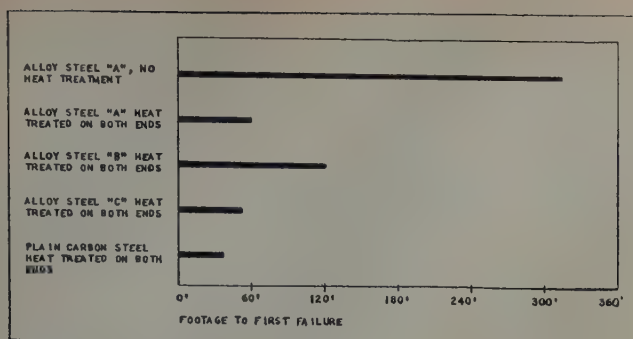


Fig. 13—Chrome molybdenum vs carbon steel rod life.

inserts. However, the large bit drills more slowly than the small one; therefore the choice of drill bit size and rock drill size for optimum drilling results depends upon trial in the particular rock in question. Long-hole drilling in hard ground to which carbide bits are particularly well adapted requires larger drills and hence larger bits.

A small jackhammer mounted on a pushfeed is a versatile piece of equipment, costing about \$550. It is a one-man machine and drills small horizontal holes with ease, provided that they are not too high; hence the outfit is particularly well suited to slashing or enlarging. Again it is necessary to make comparative tests between this type of equipment and drifters or stopers in the rock under consideration to determine which will provide the lowest cost per cubic foot excavated. Both American and Swedish manufacturers supply this equipment. The Swedish drill is sold in a package with insert rods to assure use of a drill with a light hammer blow, well within the strength limits of the carbide, thus preventing excessive carbide failure.

However, in view of the very definite improvement in uniformity and strength of American carbide, there is every reason to believe that the use of faster drills will be justified. Claims regarding air consumption should be based not on cubic feet of air per minute used by the machine but on the cubic feet of air per inch of hole drilled by the machine in a standard rock.

A closing plea should be made for greater standardization in describing mine rocks and their drilling characteristics. Such descriptions as *hard gneiss*, *hard granite*, *tough intrusive diabase* or *hard silicified sedimentary* leave much to be desired when an effort is made to correlate drilling results. Specifica-

tions should include description of rock drilled, drilling machine size, bit size or sizes, steel size, and average air pressure.

There appears to be a very definite lack of technical information dealing with rock properties of hardness and toughness applied to percussive drilling. An excellent article on "Rock Hardness as a Factor in Drilling Problems with Special Reference to the Petroleum Industry" appeared in the AIME Transactions, Vol. 190. Those who are interested in the physical properties of mine rocks are referred to the work of Obert, Windes, and Duval in Bureau of Mines RI No. 3891, 4459, and 4727, which list and describe properties of almost 150 individual rocks. The comparisons of impact toughness and abrasive hardness are of particular interest in a consideration of drillability with percussion equipment. The standard methods developed by Obert, Windes, and Duval give physical property data that is specific. A study of thin sections appears to explain why the impact toughness is high or why the abrasive hardness is high or low. The following slides of thin sections demonstrate reasons for the physical properties of these rocks.

Supplement

Pure limestone is readily drillable, as wide experience has shown and as laboratory data also indicate, see Table II. Typical Darwin tactite composed of wollastonite 45 pct, diopside 35 pct, and quartz 20 pct, is shown in Fig. 14. Grains of wollastonite, the lime silicate, form in a sub-fibrous habit to make the texture like a floor mat enclosing grains of diopside and quartz. Wollastonite sheafs are most apparent in the center and upper right and account for the evidently high impact toughness of the rock. Grains of wollastonite are from 3/10 to 5/10 mm in length while the diopside and quartz grains measure only 2/100 or 3/100 mm. By comparison, Butte vein quartz grains at this magnification would have an area 25 times the area of this slide. Drill cutting samples showed no +10 mesh fragments, but from 55 to 65 pct passed the -65 mesh screen. This indicated poor penetration and chipping, but on the other hand bit wear was not as severe as in quartz rich rocks. Both granite and Darwin hornstone, which is similar to Nevada tactite, have low impact toughness, produce coarse cuttings, and drill quite rapidly, but the presence of hard minerals results in severe abrasion of carbide, as laboratory data would indicate. Fig. 15 shows Darwin hornstone composed of 45 pct quartz of grain

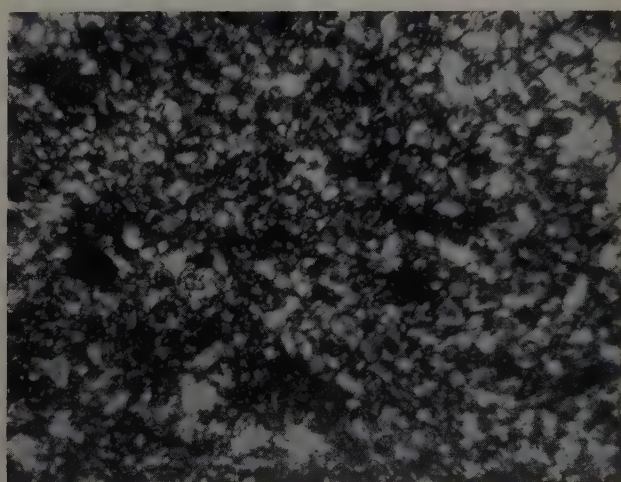
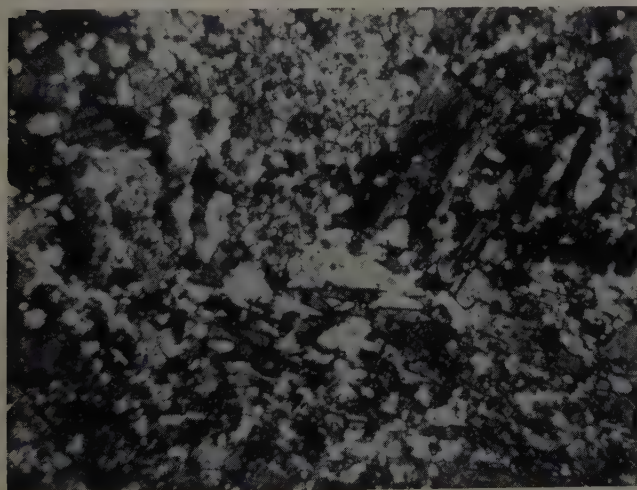


Fig. 14 (Left)—Darwin tactite, X105, and Fig. 15 (Right)—Darwin hornstone, X105.

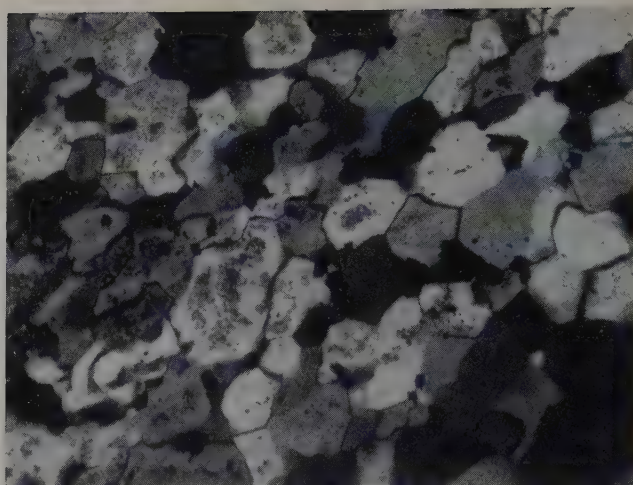
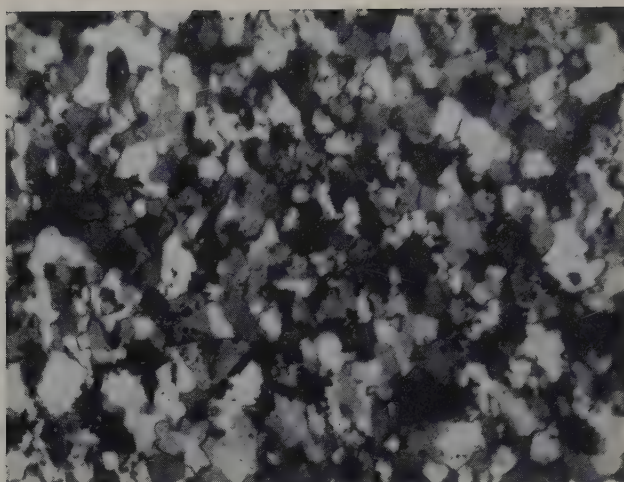


Fig. 16 (Left)—Soudan jaspilite, X105, and Fig. 17 (Right)—Swedish jasperoid, crossed nicols, X105.

size from 10 to 20 microns, 45 pct diopside with grain size from 10 to 12 microns, and 10 pct pyrite. Note that the grain boundaries are fairly uniform and not tightly interlocked. The impact toughness is not very high, drill cuttings are coarse with 20 pct +10 mesh and only 25 pct -65 mesh, and drilling speed is higher than in the tactite. However, the large percentage of fine quartz grains indicates a very high abrasive hardness, and bits are dulled in half the distance they will run in tactite.

Table II. Drillability of Various Types of Rock

Rock Type	Impact Toughness, USBM ³	Drillability, Min	Abrasive Hardness, USBM ³	Carbide Abrasiveness, In.	Cuttings, -65 Mesh, Pct
Pure limestone	8.6		10		
Darwin tactite		10*		84	55
Nevada tactite	12.0		25		
Darwin hornstone	12.6	13*		48	25
Nevada hornstone	7.1		52		
Granite, Maryland	6.9	16**	26	48**	
Jaspilite, Soudan	17.0		48		

* 2¾-in. drill, 90 psi, ⅞-in. quarter-octagon steel, 1½-in. carbide bits.

** 3½-in. drill, 80 psi, 1¼-in. round lugged steel, 2-in. steel cross bits, Rockwell C 62.⁶

Soudan jaspilite, see Fig. 16, is well known for its low drillability, and its destructiveness to drill bits

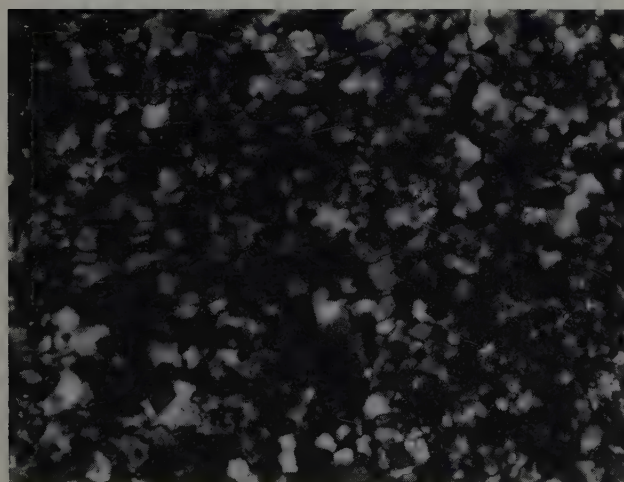


Fig. 18—Australian jasperoid. X105.

is very severe. Laboratory results from the Bureau of Mines show an extremely high impact toughness, believed to be accounted for by the sutural boundaries of the grains, which in effect lock them together as the components of a jig-saw puzzle are joined. The high abrasive hardness of jaspilite, close to that of the Darwin hornstone, is accounted for by the 96 pct fine-grained quartz composition. Charles Meyer, in charge of Anaconda's Geological Laboratory, states, "The specimen is composed of 96 pct quartz and 4 pct hematite. The grain size of both minerals ranges from 5 to 50 microns. The average semi-sutural grain shape with its high specific surface and extensive interlocking indicates high interfacial cohesion between grains."

Fig. 17 shows a thin section of a jasperoid from a Swedish iron mine. Its physical properties are unknown, but its abrasiveness to drill bits is reputed to be great. It is composed of 98 pct quartz averaging 0.10 mm in diam. Its texture suggests a lower impact toughness than the Soudan jaspilite.

The thin section shown in Fig. 18 is a jasperoid from Australian Blue Asbestos Ltd., of western Australia. It is notoriously difficult to drill. With its 98 pct quartz grains ranging from 10 to 40 microns, tightly interlocked, and with sutural boundaries similar to the Soudan jaspilite, its impact toughness and abrasive hardness are probably very high.

Further correlation of the petrographic properties, the physical properties of impact toughness and abrasive hardness, and the drillability of mine rocks should in time provide very useful data for bit selection.

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Fig. 1—Northampton quarry, operated by Universal Atlas Cement Co. This laminated rock is easily drilled, blasted, and crushed.

Cement Rock Beneficiation at the Universal Atlas Cement Co., Northampton, Pa.

by L. J. Boucher

The beneficiation process at Northampton is described and reasons are given for installing a flotation plant. The economics of running the plant, the difficulties of operation, and subsequent remedial action are discussed.

CEMENT rock from a quarry operated by the Northampton plant^{1,2} of the Universal Atlas Cement Co. has a relatively low average CaCO_3 content. To meet specifications required for cement manufacture it was formerly necessary, over succeeding years, to add increasingly larger amounts of high-lime, low-alumina rock, which was obtained from a quarry operated by the company at a point along the New Jersey-New York state border, 79 miles from Northampton. Previous to 1940, 8 pct of total kiln feed consisted of high-lime rock. From 1940 to 1943, this figure jumped to 13 pct. Today it would be necessary to use 18 pct of this high-lime to meet current specifications. Increasing transportation costs made this an expensive operation. In addition, this high-lime rock quarry was contaminated with granite and high magnesia pockets.

As there was no available high-lime rock in the Lehigh Valley, it was decided in 1936 to install a pilot plant where experimental work was done on processing local cement rock with flotation and burning the resulting product in a small kiln. This plant was in operation for six months and at the end of that period a suitable method of flotation had been established. From these studies it was also

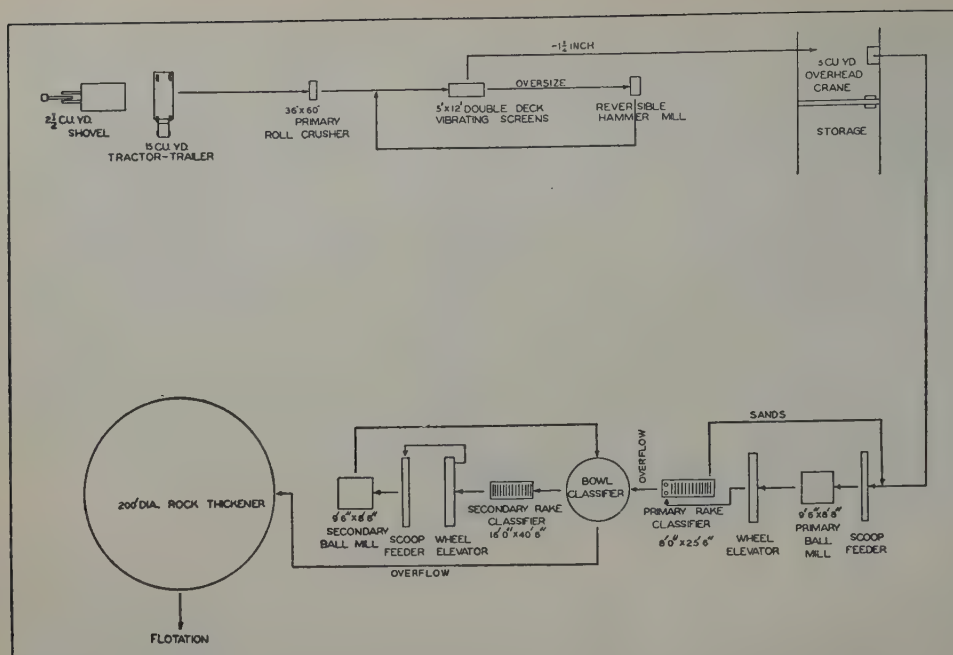
found that if the proportion of shipped-in rock rose slightly above 10 pct it became economically advisable to use the flotation process. It is the object of this paper to present briefly the operation of the flotation system at the Northampton plant. It seems advisable to comment on the raw material used in this process.

The Northampton quarry lies geologically in what is known as the Jacksonburg formation of Trenton limestone.³ In Pennsylvania this extends in a south-westerly direction from Portland, through Nazareth, Bath, and Northampton and across the Lehigh River to Ormrod. This argillaceous limestone was formed from marine deposits laid down during the early Ordovician period when a shallow sea covered most of Pennsylvania. Carbonaceous matter in the original deposit formed graphitic carbon under later metamorphic forces. From this graphite the cement rock of the Lehigh Valley derives its dark color. Most of the graphite must be removed before lime flotation can proceed.

The quartz in this rock is desirable from the chemist's point of view, as it furnishes him with a form of silica necessary in the calcining process, but its abrasive properties damage pumps and piping. Aluminum silicates of potassium, sodium, iron, and magnesium are found in the form of sericite. Calcite averages from 69 to 76 pct. Iron in the form of iron pyrites is also present and has no deleterious effect. Talc is present in minute quantities. The

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Discussion on this paper, TP 3452BH, may be sent (2 copies) to AIME before May 31, 1953. Manuscript, Feb. 12, 1952. New York Meeting, February 1952.



mineral composition of Northampton rock is shown in Table I.

The rock has been deposited in laminations, see Fig. 1, and it is therefore very easy to drill, blast, and crush. It is loaded with a crawler-type electric-shovel $2\frac{1}{2}$ cu yd dipper, Figs. 1 and 2, and transported from the quarry by means of truck trailer to a 36x60-in. single-roll crusher, giving a -8-in. product, from which it passes through doubledeck scalping $1\frac{1}{4}$ -in. screens. The oversize passes through a reversible hammermill from which it is returned to the scalping screens. The stone passing through the screens goes to storage and into raw rock grinding, which because of flotation must be done by wet process. Primary grinding is a closed circuit consisting of a 9 ft 6 in. x 8 ft 8 in. ball mill, with 4-in. grinding media and an 8 ft x 25 ft 6 in. Dorr multizone classifier with $2\frac{1}{2}$ in. per ft slope. The fines are discharged into a 27 ft 0 in. diam bowl classifier, forming a closed secondary grinding circuit, with a 9 ft 6 in. x 8 ft 8 in. ball mill with $1\frac{1}{2}$ -in. grinding media,⁴ see Figs. 3 and 4. This circuit delivers an end product of which 90 pct passes a 200-mesh screen. This slurry, containing 25 pct solids, is then pumped into a 200-ft thickener, see Figs. 6 and 7, from which it is drawn at an average 54 pct solids and pumped into a small mixing tank, where the slurry is diluted to 35 pct solids and is then stored in a turbomixer.

Table 1. Composition of Northampton Rock

Mineral	Pct
Calcite and dolomite	75.00
Quartz	8.50
Iron pyrites	.56
Graphite	.50
Sericite	15.44

The next step in the process is a separation at 30 microns in a continuous solid-bowl type of centrifuge, 54x70 in.,⁵ see Fig. 8. A peculiar situation arises when separation takes place in the centrifuge. As shown in Table II, the centrifuge feed will contain 73.17 pct CaCO_3 , the cake or +30 micron fraction will contain 75.46 pct CaCO_3 , and the effluent or -30 micron fraction will contain 71.02 pct CaCO_3 . In any grinding, the harder particles will

be the coarser, and since calcite and quartz are the hardest minerals present as compared with sericite, the centrifuge cake will contain a larger amount of CaCO_3 and SiO_2 .

The outer bowl of each centrifuge, see Fig. 8, rotates at an average speed of 320 rpm, while the inner bowl rotates at 304 rpm, maintaining a 19 to 20 speed ratio with the outer bowl. At this speed a 30-micron separation takes place. Naturally, a variation in speed will change the dividing line between particle size separation. It is also important that the specific gravity of feed be set at 35 pct solids, as a more viscous mixture results in poorer separation due to difficulties of removing the smaller particles with insufficient water.

The cake with the 75.46 pct CaCO_3 is stored for blending without further processing, as the flotation process will furnish a product of much higher lime concentration, and the blending with other products results in a kiln feed of required composition.

Each centrifuge produces on an average 56.2 tons per hr if its separation is efficient. Presented in Table III are the cumulative percents finer than micron sizes of the feed and the discharges.

This operation is continuous, except on weekends, when the whole flotation system is shut down. Before stopping, it is necessary to wash and remove any deposits, which harden if left standing, causing an unbalanced load and considerable vibration when the centrifuge is started. The main difficulty experienced in this phase of the operation was that in the cone on which the screw was mounted and into which the feed pipe discharges the rinse water did not drain completely. A number of 1-in. diam holes were drilled along the circumference of the cone to correct this condition. There has been no trouble since caused by vibration.

After the effluent leaves the centrifuge, it is diluted to 17 pct solids and fed into a turbomixer, from which it is pumped into a constant head feed tank for the flotation cells. The flotation system, see Fig. 9, consists of four parallel rows of eighteen 66x66x27 in. cells.^{6,7} These are of the mechanical-agitation type, each cell being equipped with a motor-driven vertical rotor. The most efficient speed of these rotors was found to be at 420 rpm.

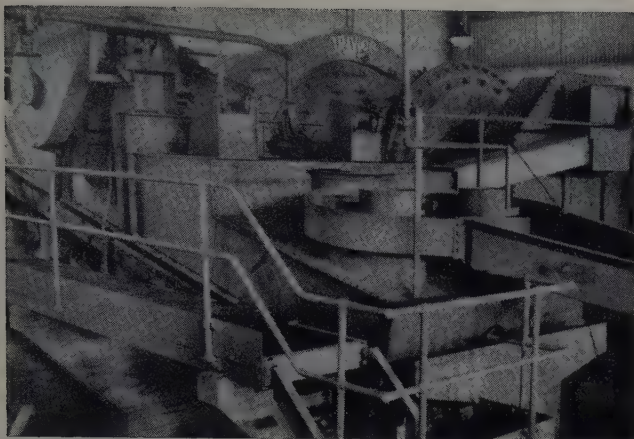


Fig. 3—Grinding circuits utilize multizone classifiers.

Skimmers placed along the sides of the cell group accelerate the froth removal. Average production rate, per row of 18 cells, is 10½ tons of solids per hr, or 0.148 tons per hr per cu ft of cell volume. Power requirement is 4.5 hp per cell. There is very little maintenance, as both stator and rotor bars are rubber-covered and require little attention.

In operation, a carbonaceous concentrate is taken off in the first three cells of each row. This is relatively low in lime and quite high in micaceous and clay materials. The object of eliminating this material at this point is to facilitate the ensuing flotation of lime and to save on the consumption of reagents which would be absorbed by this carbonaceous material. Diesel oil is used as the collector and methyl isobutyl carbinol as the frother.

Table II. Analysis of Centrifuge Feed, Wt Pct

Com- pounds	Feed	Cake, + 30 Micron	Effluent, - 30 Micron
SiO ₂	14.28	13.80	14.80
Al ₂ O ₃	4.98	3.86	5.96
Fe ₂ O ₃	1.60	1.50	1.68
CaCO ₃	73.17	75.46	71.02
MgCO ₃	4.42	4.52	4.23

Normally the amount of collector reagents added is held constant and the frother is varied to maintain the carbonaceous concentrate at 58 to 60 pct calcium carbonate at the first three cells. If this car-

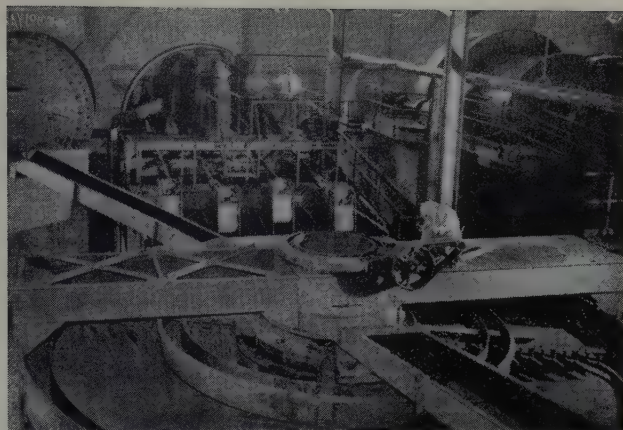


Fig. 4—Fines overflow 27-ft bowl classifier to rock thickener.

bonaceous froth goes below 58 pct CaCO₃, the amount of carbon removed is too low. If it goes above 60 pct, too much CaCO₃ is being removed, and this will be reflected in a lower efficiency. Samples are tested regularly, for care must be taken not to draw off calcium carbonate excessively. The carbon fraction joins with the final rejects and is discarded.

Table III. Centrifuge Performance, Cumulative Percents Finer Than Micron Sizes of Feed and Discharges

Micron	Feed	Cake	Effluent
44	88.7	83.7	100
30	72.2	64.3	100
20	64.9	53.2	94.6
15	57.8	42.7	93.3
10	48.8	34.5	83.4
8	45.2	31.7	77.4
7	43.1	30.8	73.2
6	40.6	29.7	69.8
5	38.3	28.9	65.9
4	35.5	27.8	60.2
3	34.7	27.0	55.0
2	29.6	25.5	48.8

In the fourth cell, the beginning of the high lime concentrate separation, calcium lignin sulphonate is added to disperse the remaining carbon. Anionic reagents, oleic acid, and saponified tall oil are used from this point on, saponified tall oil in Cells 4 to 14, and oleic acid in Cells 4 to 18. Thus oleic acid alone is used in the last four cells, because it is the better collector and is required when the recover-

Fig. 5—Slurry is drawn from a 200-ft rock thickener at an average 54 pct solids. In the mixing tank it is diluted to 35 pct solids, after which it is stored in a turbomixer.

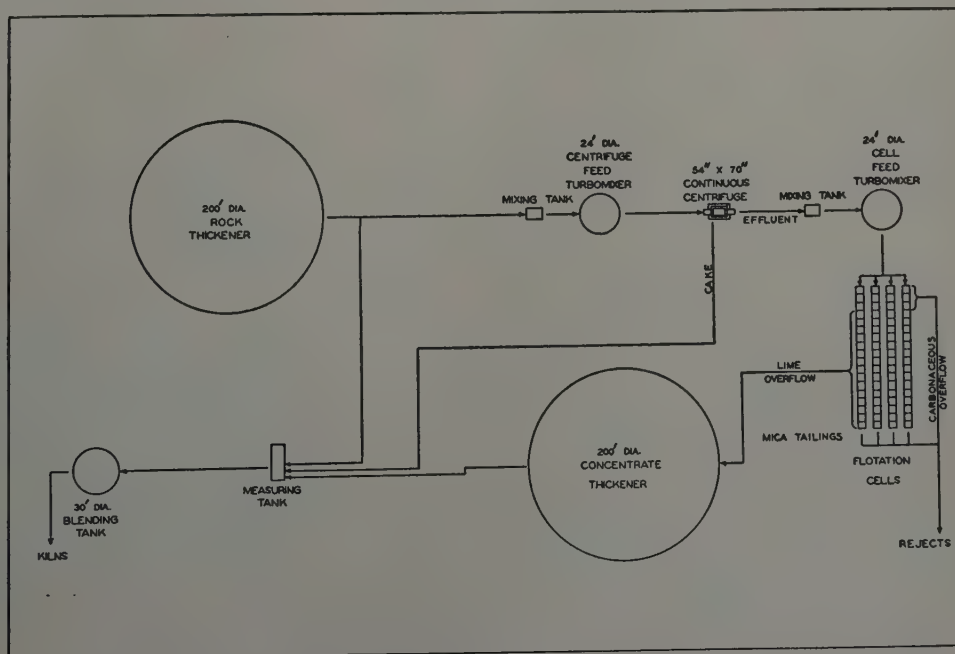




Fig. 6—The 200-ft thickener receives rock slurry from the grinding circuit. In left foreground is a small clear water overflow tank.

able calcite is as low as it is in these cells. As tall oil is the less expensive of the two, a ratio of 71 pct tall oil to 29 pct oleic acid is used overall. Reagent regulation requires careful adjustment, see Table IV.

These anionic reagents react with strong cationic minerals, such as calcite, at the surfaces of the particles, while with the other minerals present there is no such reaction. This reaction forms a compound which is insoluble in water and buoyant, rendering the particles water-repellent. Myriads of tiny air bubbles are whipped throughout the cells by me-

Table IV. Regulation of Reagent Consumption

Reagent Consumption	High Lime Concentrate, Lbs Per Ton
Diesel oil	.161
Methyl isobutyl carbinol	.231
Oleic acid	.314
Saponified tall oil	.793
Calcium lignin sulphonate	.953

chanical agitation of the rotor, so that the air bubbles attach themselves to water-repellent particles and raise them to the surface, see Fig. 10. Normally the bubbles burst at the surface and drop their mineral load, but a froth is maintained at the surface primarily by addition of the frother, methyl isobutyl carbinol, in the carbon cells holding the mineral until a skimmer removes the froth. A pH regulation of reagents is not necessary. Limestone slurries are normally at the optimum pH for best separation, which is about 8.2.

Flotation of cement rock is affected particularly by temperature, especially in winter, by grade of rock, variations in fineness, and amount of solids. In the case of exceptional changes in the grade of rock, changes in the amounts of collectors must be

Table V. Analyses of Flotation Plant Products

Compound	Cell Feed	High-Lime Concentrate	Carbon Rejects	Mica Rejects*	Combined Rejects
SiO ₂	14.80	6.72	15.36	47.24	39.32
Al ₂ O ₃	5.96	3.00	6.96	15.73	13.59
Fe ₂ O ₃	1.68	1.30	4.00	2.68	2.43
CaCO ₃	71.02	84.06	59.87	21.61	31.77
MgCO ₃	4.23	3.27	5.14	7.31	7.13

* Mica rejects are the underflow from the last cell.

made. The percentage of CaCO₃ in the lime concentrate froth can be increased by decreasing the amount of frother or collector. This, of course, decreases total volume of concentrate derived and increases the amount of rejects per ton of cell feed. Calcium carbonate titrations are made regularly on samples of high-lime concentrate from each cell and on underflow rejects from the last cell.

The high-lime concentrate taken off Cells 4 to 18 inclusive is slightly above 84 pct CaCO₃ and this



Fig. 7—Raw rock thickener, left; flotation cell house and blending tanks, center; and 200-ft concentrate thickener, right.

flows by gravity into the high-lime 200-ft concentrate thickener which serves as a storage tank and water remover. The carbonaceous concentrate taken off Cells 1 to 3 combines with the underflow at the end of the cell row and is discarded by pumping to an abandoned quarry. Overflow water from the high-lime concentrate thickener is retrieved for use again in the flotation system.

Representative analyses of the flotation plant products will be found in Table V.

Data of a typical day's operation, using average-grade quarry rock, is as follows. On a dry tonnage basis, 2680 tons of cement rock are fed to the two centrifuges, of which 1340 tons are of the +30 micron fraction in the centrifuge cake, and the other 1340 tons of -30 micron fraction in the centrifuge effluent that constitutes cell feed. Of this latter 1340 tons, 1008 tons are high-lime concentrate at 84 pct CaCO₃ and the remaining 332 tons are rejects at 31.77 pct CaCO₃. Of the total volume of rock fed to the centrifuges, 87.6 pct CaCO₃ is recovered by the centrifuges and cells. Of the total volume of rock fed to the cells, 75.2 pct CaCO₃ is recovered. Varying with the quality of rock, see Fig. 8, an average 33.8 pct of cement rock is drawn directly to blend tanks for blending along with centrifuge cake and high-lime concentrate. With these materials available at all times, together with sand and pyrite cinders, production of any type of cement can be started in a short time. In every case these materials can be proportioned to give satisfactory kiln efficiency and make cements meeting all requirements of all specifications and consumer demand. If only one kind of cement is made and rock from thickener is quarry average rock, the amount of rejects discarded is 8.2 pct of total rock quarried. When one kind of cement only is made, if grade of



Fig. 8—The outer bowl of each continuous 54x70-in. centrifuge rotates at 320 rpm, the inner bowl at 304 rpm, effecting a 30-micron separation.

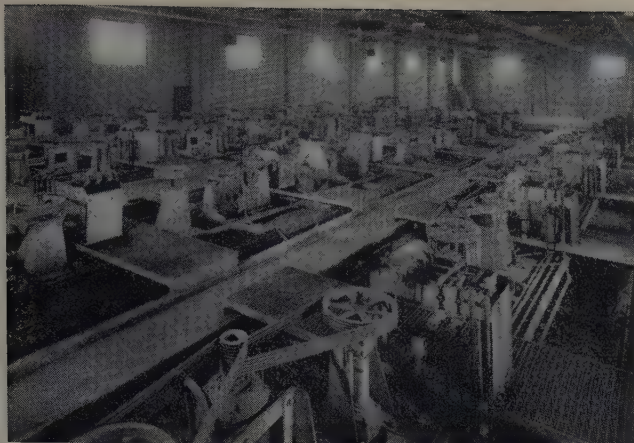


Fig. 9—A view of the flotation system, four parallel rows of 18 cells each. Mechanical-agitation cells are 66x66x27 in.

rock is above 73.2 pct CaCO_3 , the system's efficiency improves proportionately. If other types of cement are made or the grade of rock falls below 73.2 pct, the percentage of rejects increases.

Personnel required for one 8-hr shift includes one operator and one helper, one pumpman, one pumpman helper, and one laboratory control man. These men handle centrifuge operation, flotation, blending, and pumping to kilns. Flotation normally operates 24 hr per day, 5 days per week. When high-lime concentrate thickener becomes full, flotation operations must be curtailed. Grade of rock, amount and type of cements made, and kiln consumption are the governing factors.

This entire operation, centrifuging and flotation, occurs in a building known as the Cell House, in the basement of which flotation reagents are prepared in six make-up tanks. These tanks are complete with steam coils and agitating apparatus. The Cell House contains its own oil-fired heating plant for this purpose.

As previously stated, it is very important to control the density of material to centrifuges. Formerly this was done by weighing slurry samples from the mixing tanks every half hour and adjusting the water flow accordingly. As the specific gravity of the thickener slurry varied from 40 to 60 pct, with a general average of 54 pct, it is evident that this method was not very accurate. A Foxboro Stabilog recording liquid density controller, Model 40, with a density range of 1.10 to 1.40 sp gr, complete with an operated Foxboro butterfly control valve, was installed to dilute the thickener slurry automatically to 35 pct solids, or 1.283 sp gr. Slurry and diluting water pipes are located at one end of the mixing tank in which this is done and a sampling pipe for the density control instrument is located at the other or discharge end. This sampling pipe is 2 in. in diam and has a minimum number of bends which must be of a large radius to prevent clogging. It was necessary to install a valve in the bottom of the sampling chamber of the density control unit to provide a self-cleaning outlet. The density control instrument is very susceptible to dirt in air. After some experimentation it was found best to use two filters in series, the first having a radial fin filter insert to remove moisture and larger airborne dirt particles, and the second containing a series of absorptive discs mounted over a central supporting tube to remove the remaining finer particles.

Another interesting accessory to the flotation system is the Clarkson reagent feeder consisting of a constant speed driven disc with a number of small



Fig. 10—Air bubbles whipped throughout the cells attach to water-repellent particles and rise with them to the surface.

capacity buckets attached. With 9/16x3/8-in. buckets used, the rate of feed can vary from 0 to 11 cu cm per min.

The handling of slurry brings up several problems. One is the constant cleaning with water of piping that has been used to transport slurry. If this is not done, a sediment is deposited each time the pipe is used, and soon the pipe will have to be replaced. When tees have one unused leg that leg is soon clogged up. It is a practice to place valves as near as possible to the point of intersection. Valves present another problem. A pinch valve has been experimented with and has been found to outlast a standard valve by a ratio above 3 to 1.

There are, necessarily, a large number of centrifugal pumps which require considerable attention. Because of the abrasiveness of the slurry, there is an excessive amount of wear on pump casings, impellers, and suction plates. After considerable metallurgical experimentation, pump parts made of Ni-Hard have been found superior to those made of other metals.

It might appear that use of the flotation process for beneficiation of low-cost cement rock as required for cement kiln feed would be complicated and uneconomical. Rather, with reasonably intelligent operators, this operation soon becomes routine. While initial cost does increase capital investment, the process has proved economically successful at the Northampton plant and other plants where this type of raw material is available.

Acknowledgments

The author is indebted to the following collaborators: L. G. Sprague, Research Chemical Engineer, Separation Process Co., Catasauqua, Pa., R. F. Moore, Chief Chemist, and William Feenstra, Project Engineer, both of Universal Atlas Cement Co., Northampton, Pa.

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Some Features of Current Mining Practices At Kerr-Addison Gold Mines, Ltd.

by W. S. Row

This mine is operated at 4000 to 4500 tons daily through a single shaft, with one rock hoist and one service hoist. Latest shaft construction is concrete with wooden dividers. Economics of drifters and air-leg drills are compared, as well as costs of shrinkage and sub-level stoping. Economy of large haulage equipment and underground crushing is shown.

THE property of Kerr-Addison Gold Mines is located in the Larder Lake district of Ontario about 3 miles from the Quebec boundary on the highway between Kirkland Lake and Noranda. It is roughly 320 miles due north of Toronto and 365 miles north of Buffalo, N. Y.

The property was staked in 1906 and after several development efforts finally came into continuous production at 500 tons per day in 1937. In 1939 the daily tonnage was increased to 1200 and in 1941 to 2100 tons. At the end of the war, because of the shortage of labor, tonnage was down to 1100 tons per day, rising to 2200 in February 1947. A further plant expansion inaugurated in 1946 was completed by December 1948, bringing the capacity of the mine and mill to a production rate of 4000 tons per day. Current production, with 880 men employed, is 4400 tons per day. Changes in mining methods and equipment, required by the recently completed expansion program, have yielded interesting comparative results, some of which are described in this paper.

Hoisting of ore and waste from the mine, as well as all service facilities for handling men and supplies underground, are provided for in a single vertical five-compartment shaft. Ore and waste are hoisted in 12½-ton skips in balance by a 14-ft diam hoist with drums 115 in. wide, grooved for 2⅞-in. wire ropes. This hoist, with a total rope pull of 85,000 lb, was designed to operate at a speed of 2600 ft per min to a depth of 4000 ft, hoisting 15 tons of ore in each skip. It is amplidyne-controlled and powered with dual drive dc motors of 2250-hp each, through flexible couplings to two pinion shafts, one on each side of the main gear. Each motor has a peak load rating of 4500-hp at point of greatest duty. Direct current power to the hoist motors is supplied from a motor generator set consisting of a

5000-hp synchronous motor driving two 1750-kw dc generators.

The loading pockets used so far are at depths of 1500 and 2725 ft respectively. Consequently the present hoisting speed is 2000 ft per min and the skips capacity only 12½ tons. Now that the shaft has been deepened the new pockets at 3934 ft will be increasingly used so that the speed and the skip size will be increased up to original specifications as required.

Some difficulty in meeting the required hoisting cycle is expected because of the shape of the skips. As now in use the cross-sectional dimensions of the

Table I. Time Analysis of Skip-Hoisting Operation, 1951

Item	Pct
Hoisting ore and waste required	73.8
Maintenance	7.9
Inspection	2.0
Service	9.3
Operating delays	7.0

inside of the skip are 4 ft 6 in. x 3 ft 8 in. To hold 12½ tons, therefore, they are 14 ft 10 in. deep. Since the skips are the conventional overturning or Kimberley type, considerable hoisting time is lost in retardation in entering the headframe dump and in acceleration when leaving. It is expected that to obtain the specified tonnage from this hoist when the skip size is increased to 15 tons it may be necessary to resort to bottom dump skips. This type of skip could reduce considerably the present excessive acceleration and retardation times.

Analyses of the skip-hoisting operation are made monthly, and every effort is made to reduce controllable delays. For a nine-month period in 1951 such an analysis shows that the hoist was in operation 92.5 pct of the available time, which was allotted as shown in Table I.

Overall hoisting costs, cage and skip, for the year 1951, averaged \$0.168 per ton of ore milled.

The mine is serviced entirely by means of a single cage, capacity 35 men or 15,700 lb, operating in con-

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Discussion on this paper, TP 3487A, may be sent (2 copies) to AIME before May 31, 1953. Manuscript, Jan. 21, 1952. New York Meeting, February 1952.

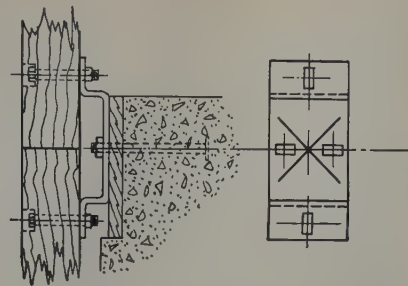
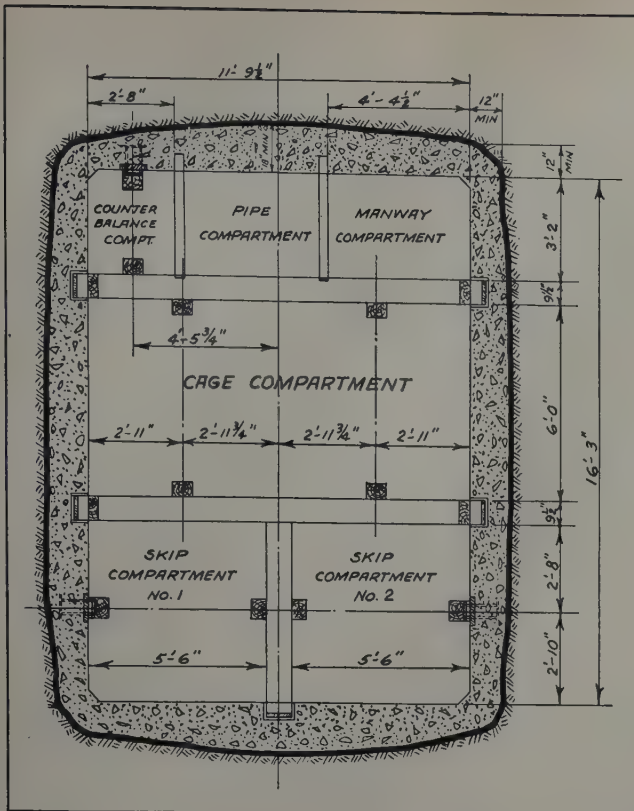


Fig. 1—Plan of concrete ring with timber dividers. Concreting was chosen for shaft support because less maintenance, reduced fire hazard, less excavation, and better ground support offset immediate cost advantage of timbering. Upper right, guide brackets are attached to concrete.

junction with a counterbalance. The cage is at present single-decked, with provision made to add a second deck below. Made of aluminum, it is 10 ft 9 in. long by 4 ft 11 in. wide, inside measurement, and operates on four wooden guides. The cage hoist is a double-drum, single-clutched, electrically driven hoist, designed to operate at 2000 ft per min to a depth of 4150 ft. The designed rope pull is 33,100 lb. The drums are 10 ft in diam by 72 in. and grooved for 1½-in. ropes. Originally the hoist was driven by an 800-hp ac motor, operating the hoist at 1280 ft per min. Recently another 400-hp motor has been added, driving through a pinion on the other side of the main gear and bringing the hoisting speed up to 2000 ft per min. The control of the two motors has been arranged for dynamic braking. To date this has been very effective and has eliminated the heavy duty previously required of the brakes, even though the hoist is now operating at a much higher speed.

The No. 3 Shaft

The shaft, dimensions of which to outside of timber are 17 ft 6 in. by 13 ft ½ in., is timbered with 10x10 in. B.C. fir sets at 6-ft intervals to a depth of 2800 ft, where sinking operations were stopped in 1942. When sinking was resumed early in 1950, consideration had been given to the relative merits of timber vs concrete as a means of providing adequate ground support. It was finally decided to use a series of concrete rings 5 ft deep at 7-ft centers, leaving a 2-ft space between rings, see Fig. 1. Timber dividers were to be cast in the top of the concrete rings, brackets to be attached to the concrete where necessary to support conveyance guides. The following points were examined: 1—The overall cost would be higher than with timbered sets. 2—Progress would be slower. 3—Since there were no company shaftmen with experience in concreting, a complete crew would have to be trained.

Other factors offset these considerations: 1—The gradually increasing cost of B.C. fir narrowed the margin between the cost of timber and the cost of concrete. 2—If the amount of timber in the shaft were reduced, maintenance in later years would also be reduced and the fire hazard lessened. 3—Based upon previous experience with timbered sets, standard practice in weak ground was to excavate sufficient ground at the north and south ends of the shaft to provide room for jacket sets to release pressure on the end plates. This would not be necessary with concrete, resulting in a saving of 9 tons of rock excavated per vertical ft of shaft. 4—Concrete provides uniform support over a wide area, not the localized support of blocking from timbered sets. This is important in soft ground, which flows around the end of the blocking. In blocky ground the more uniform support greatly reduces overbreak if the concrete is kept close to the bottom.

Plans to resume sinking were based on a shaft concreting operation at another mine. In April 1950 actual sinking was resumed, on a 2-shift per day basis, under a bulkhead at the 2800-ft level.

Since the concrete was to be carried to within 10 or 12 ft of the bottom, it was decided to adopt a cut and square drilling pattern to minimize the danger of damaging the concrete and to allow as much time as possible after the concrete was poured before blasting. Tungsten carbide-tipped rods and bore pluggers, 2 9/16 in., were used for all drilling.

Mucking was done by hand, as the various types of mechanical muckers were not readily adaptable to this particular job. The mucking performance per man was far below what had been anticipated, so in October the drilling pattern was changed to that of benching, see Fig. 2. Immediately there was a decided improvement in the breaking and mucking performances, with the crew mucking one bench and drilling and blasting another in one shift. Approximately 75 to 80 pct of the muck could be scooped into the buckets.

Initially three 21-cu ft sinking buckets were used. On Nov. 27, 1950, when the shaft had reached a depth of 3250 ft, larger buckets with a capacity slightly in excess of 1½ tons each were installed to lessen the hoisting delay, which was becoming progressively greater as the depth increased. During the benching procedure in November 1950, when muck was being scooped into the smaller buckets, the performance was 6.15 tons mucked per man

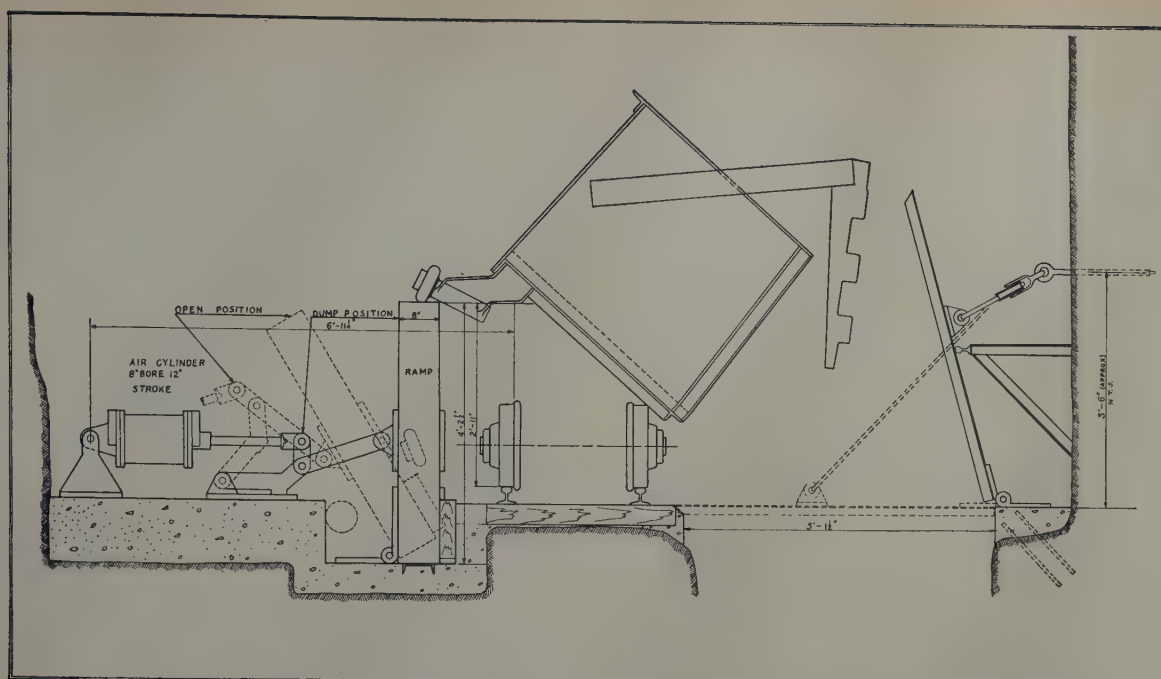


Fig. 4—Dumping arrangement used incorporates air-operated horizontal steel door to close opening, confine dust. Hinged ramp for dumping Granby-type cars can be tilted to pass cars without dumping.

concrete were used and to this mix were added $\frac{1}{2}$ lb of pozzolith and 1 lb of calcium chloride per bag. Test cylinders were taken from every third ring and tests revealed a compressive strength in excess of 3500 lb per sq in. after 28 days. Excavation was sufficient to allow a minimum wall thickness of 12 in. of concrete in the ring and 18 in. in the center of the north and south walls, which are parallel to the schistosity.

Since the entire crew could not be used effectively for cutting stations, this operation was deferred until sinking was completed. Also it has been demonstrated, in this operation and others, that interruption of the sinking cycle to cut stations at each level is reached lowers the efficiency of both the sinking and the station cutting operations. In October 1951, with crews of 8 men on a 3-shift basis, work was started on station excavation. That part of the station adjacent to the shaft only is being excavated now to provide sufficient room to install the shaft station timber and concrete. Upon completion of all the stations to this stage, the skips and service cage can be operated to the 3850 level, at which time the excavation of the stations to full

finger raise from the pass leads up to the main crosscut to provide a dump for the ore trains. Over this raise is a horizontal steel door hinged on one side and opened by an air cylinder. This door covers the opening when no trains are dumping and confines the dust stirred up in the operation of the pass. At each dump also is located a hinged ramp which is locked in place while in use for dumping the Granby-type cars, see Fig. 4. When it is necessary for these cars to pass without dumping, an air cylinder tilts the ramp sufficiently to allow clearance for the cars to pass.

Ore Pass System and Crusher Station

At each level a control chute is installed in the ore pass system located in a 10x12 ft station at track elevation, see Fig. 5. Access to this station is by the aforementioned small drift, from the main crosscut, equipped with a dust-tight steel door. The throat opening of this chute is 8 ft wide by 6 ft normal to the flow of the ore. The chute is constructed entirely of concrete and is lined with 3-in. abrasion resistant steel plates 2 ft square. These plates are attached to the concrete by means of $1\frac{3}{8}$ -in. bolts into inserts cast in the concrete, see Fig. 6. Thus these plates can be easily replaced when worn out. Formerly 1-in. diam bolts were used but were found to be too small. Each 3-in. plate weighs 450 lb, and for one chute $26\frac{1}{2}$ tons of steel liners are required. Flow of the broken ore through the chute is controlled by a chain curtain made up of links of 3-in. anchor chain. The total weight of the chain curtain is $5\frac{1}{2}$ tons; it is operated by a 14-in. diam air cylinder with an 8-ft stroke and controlled by a 3-way valve located outside the door in the connecting drift.

At the bottom of the ore pass system just below the 2500-ft level is located the crusher station in which the bulk of underground crushing is now done, see Fig. 7. The jaw crusher, 42x54 in. in size, is fed by a vibrating type of feeder and grizzly with 6-in. openings. Ore to the feeder is controlled by a chain gate which is operated by an air cylinder. The

Table II. Comparative Shaft Sinking Data

Item	Cut and Square, April- Oct. 1950	Benching, Jan.- Sept. 1951
Avg no. men per crew	10.6	11.7
Ft advance per man shift, break only	0.53	0.74
Tons mucked per man shift	4.88	6.73
Man shift per concrete ring	18.38	13.04
Total man shift per ft advance	10.53	7.40
Powder, lb per ft advance	36.6	39.6
Maximum advance per month, ft	68.0	98.0

size will be finished and the initial development work will start.

An ore pass system continuous from the top of the mine to the bottom is located between the shaft and the ore zone. The system is composed of a series of connected 8x12 raises driven at 65° from short drifts off the main crosscuts. Below each level a

crusher and feeder are totally enclosed by a dust-tight metal housing connected by exhaust ducts to a dust filter. A door with an armor plate glass window allows access to take wood out of the crusher. While the rock in the vicinity of the crusher station was fairly competent, part of the station roof required support. To insure a minimum of maintenance in the future, the entire roof was supported with reinforced concrete, see Fig. 8. The walls were likewise supported with heavily reinforced horizontal concrete rings and vertical columns. The walls at each end of the station were concreted solidly and the other two walls were covered by gunite between the columns and rings.

Table III. Comparative Shrinkage Stoping Data

Year	Type of Drill	Tons Broken	Tons Per Man Shift	Powder Per Ton, Lb
1950	Drifter	739,000	45.2	0.38
1950	Air-leg	147,000	60.0	0.38
1951	Drifter	355,000	39.4	0.41
1951	Air-leg	543,000	55.8	0.36

Excavation of the crusher station was started at the bottom from the previously driven ore pass raise and mined out by shrinkage methods. Concreting of the roof and the reinforced rings and columns were carried out progressively from the top down by drawing the broken rock down as required. Secondary breaking is reduced to a minimum, and costs thereby held down if the ore is introduced to crushers in the coarsest possible state. However,

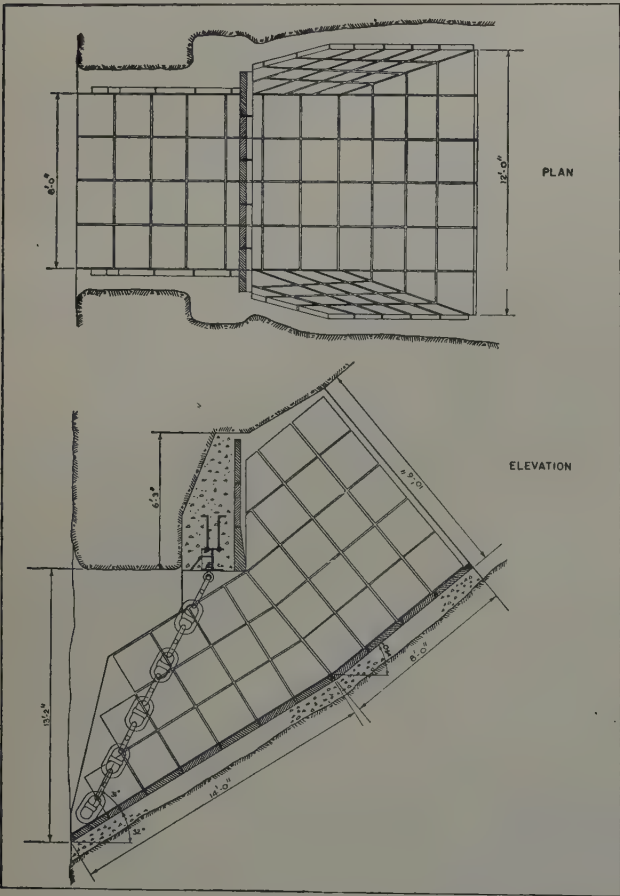


Fig. 5—Diagram of control chute installed in ore pass system at each level.

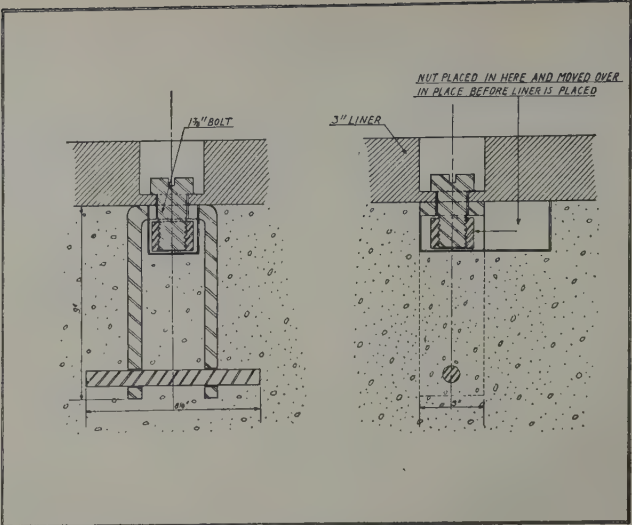


Fig. 6—Mounting for abrasion-resistant steel inserts in a concrete chute.

very coarse ore does not flow freely through control chutes and measuring pockets and into skips. The handling of large chunks of ore in skips results in lowered hoisting efficiency and higher maintenance costs of both skips and shaft as compared to an operation where crushed ore is hoisted. For these reasons primary crushing has always been done underground at Kerr-Addison.

Stoping methods in use during 1951, see Table III, accounted for the following percentages of ore broken during the year: shrinkage 77 pct, sub level 16 pct, cut and fill 2 pct, and pillar recovery 5 pct.

Shrinkage Stoping

Shrinkage stoping has accounted for the greater percentage of mill feed to date. Wide orebodies on the upper levels have been mined by a series of transverse stopes, but general practice is to lay out the stopes with their greatest dimension along the strike of the orebody. Good ground has made it possible for widths up to 150 ft to be mined by shrinkage stoping operations. Normally orebodies under 30 ft in width are developed at 150-ft vertical intervals; those wider than 30 ft are mined through a 300-ft vertical lift, see Fig. 9.

A regular pillar system provides temporary support to the walls during mining operations. These pillars are mined by long-hole drilling when the stopes on each side are pulled empty or by cut-and-fill methods under certain conditions.

The year 1951 has seen all drifters replaced in stopes with air-leg drills using tungsten carbide-tipped rods, the bits of which are chisel-shaped. As was anticipated, the changeover resulted in an appreciable increase in tons broken per man shift.

The lighter equipment has materially reduced the frequency of accidents caused by strains through carrying heavy equipment over relatively rough muck surfaces in stopes.

Data over a period of nearly 2 years, see below,

Period	Frequency Per 100,000 Shifts		Severity Per 100,000 Shifts		Rating	
	Drifter	Air-Leg	Drifter	Air-Leg	Drifter	Air-Leg
1950 & 1951	1.45	0.42	47.7	10.0	69.0	4.2

shows that not only is the accident frequency lower with air-leg drills but that the accidents which do happen are less severe.

Short period delay blasting caps are now used in the majority of shrinkage stopes. This appears to have resulted in better fragmentation and in less lost time from cutoff holes, but as yet insufficient data are available to indicate any definite decrease in secondary breaking costs.

Sub-level Mining

Sub-level mining methods have been applied chiefly to flow-type orebodies where the ore outline is more regular than in the carbonate zones and where walls are sufficiently strong to stand over a long period of time. The regular pillar system used with shrinkage mining lends itself to sub-level mining as well. All drilling is done from hanging wall and foot wall drifts, holes being laid out in a radial pattern with rings at 7-ft intervals, see Fig. 10. Within the radial pattern, tonnage per foot of hole drilled equals 3.0, but because of the slightly smaller burden in the slot area and also because of added holes on the walls, the overall tons per foot of hole averages 2.54. All blasting is done with 75 pct, 1¼x8 in. Dygel. Short period delay blasting caps are used. Average powder consumption is 0.25 lb per ton.

Initial sub-level mining was done with diamond drills, working from sub-levels at 75-ft intervals. Sectional steel rods with 1¼-in. tungsten carbide bits were tried under similar conditions, but company experience had proved that 50 ft was the economic limit to which a hole could be drilled. Consequently the sub-level interval was reduced to 50 ft with certain definite advantages: 1—No holes longer than could be drilled economically with sec-

tional steel were necessary. 2—The shorter interval permitted closer geological control, resulting in a more complete extraction of ore where irregularities occurred in the ore outline. 3—Slot raises could be driven from sub level to sub level without timber. 4—The proximity of the first sub level to the box-holes permitted the sill cut to be mined by long hole drilling. 5—While the maximum distance is maintained between the toe of adjacent holes in any one ring, the efficiency of the collar of the holes is increased slightly as the sub-level interval is decreased. 6—The 50-ft interval fitted in with the 150-ft main level interval, placing every third sub level at a main level.

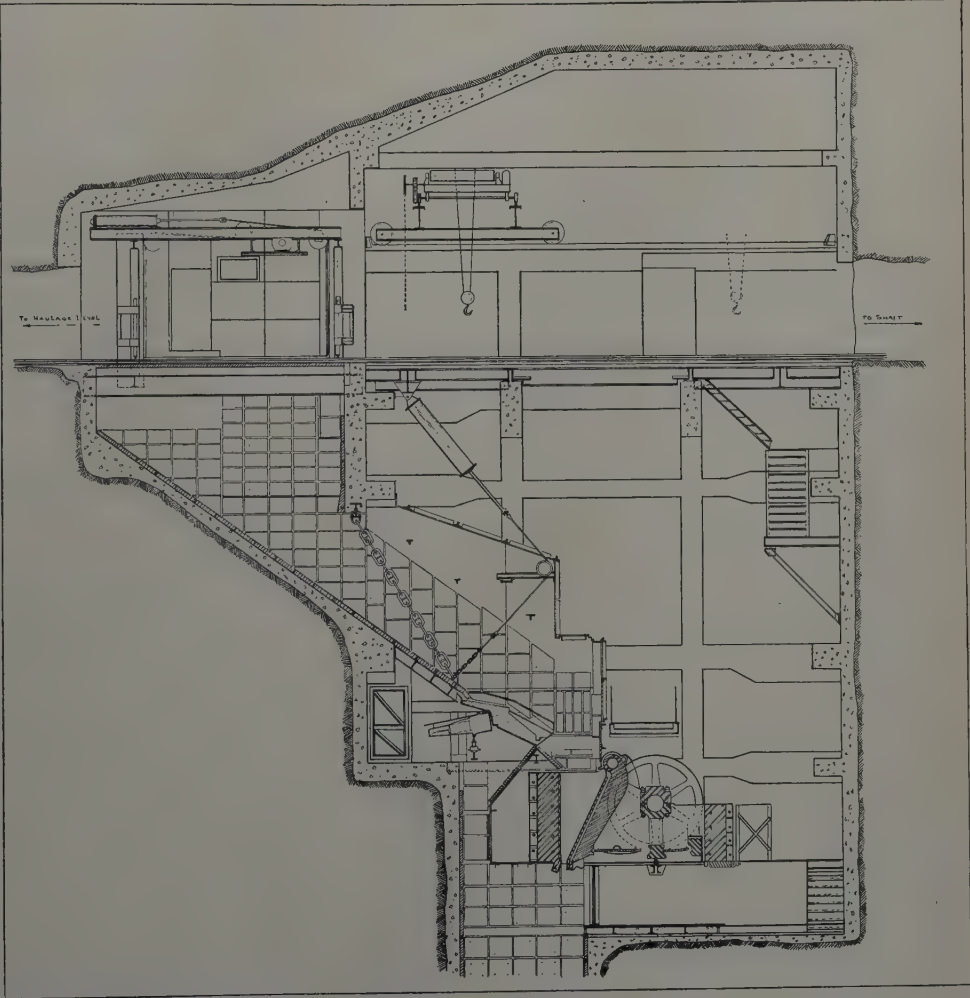
Table IV. Comparative Sub-Level Stope Drilling Data

Jan.-Nov. 1951	Footage Drilled	Labor, \$	Rods, Bits, Drill Mtee., Comp. Air, \$	General Ex- pense, \$	Total Cost Per Ft., \$
Diamond drill	97,516	0.29	0.46	0.18	0.93
Sectional steel	20,855	0.26	0.23	0.17	0.66

Diamond drills are now being replaced in sub-level stopes by 3½-in. drifters with automatic feed, 1⅞-in. hexagonal sectional drill rods in 30 and 60-in. lengths, and 1¼-in. tungsten carbide bits. Couplings are 3 in. long and 1⅝ in. OD, with reverse buttress thread.

A complete conversion to sectional steel will not be possible, since to obtain geological information from two to three holes are diamond drilled on every third ring. This may account for as much as 10 pct of the entire footage drilled.

Fig. 7—Underground crushing station at 2500-ft level at bottom of ore pass system. The 42x54-in. jaw crusher is fed by vibrating feeder.



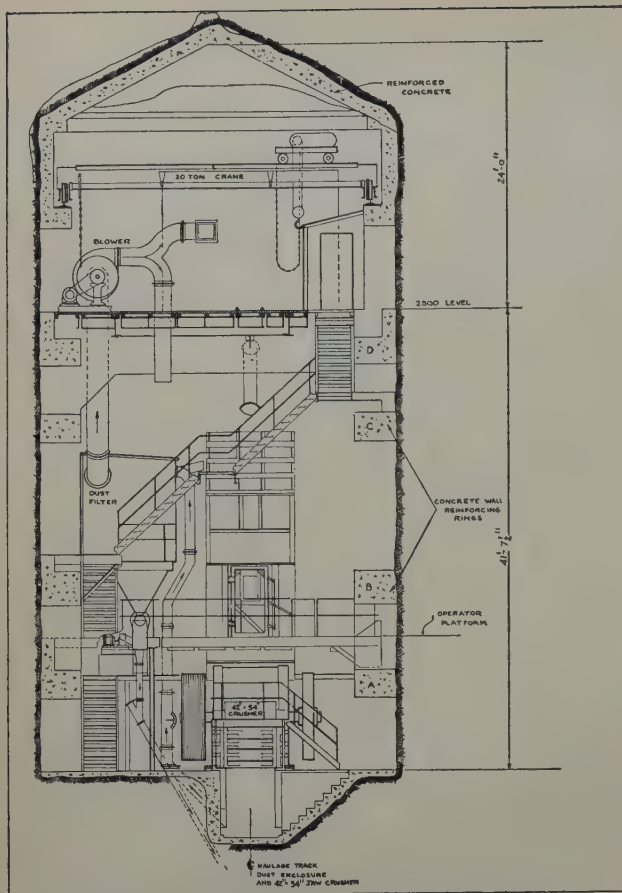


Fig. 8—Roof of underground crushing station at 2500-ft level is supported by reinforced concrete. Walls have heavy concrete rings and columns.

Table IV offers a comparison of sub-level stoping drilling costs by diamond drill and sectional steel from January to November, 1951.

Of interest is a comparison of the cost of mining a block of ore by 1—shrinkage mining, with air-leg drills and tungsten carbide bits and drifters using steel detachable bits, and 2—sub-level mining, with diamond drill and sectional steel.

To obtain a fair comparison, a block of ore 50 ft wide, 100 ft long, and 300 ft high was assumed, and to this the above mining methods were applied,

Table V. Cost Comparison for Mining Block of Ore by Various Methods

Mining Method	Stope Dev., \$	Min-ing, \$	Total	Total Tons	Cost Per Ton, \$
Shrinkage, air-leg	9,561.00	68,630.72	78,191.72	106,190	0.74
Shrinkage, drifter	9,561.00	78,131.52	87,692.52	106,190	0.83
Sub-level stope, diamond drill	31,243.37	55,454.88	86,698.25	105,712	0.82
Sub-level stope, sectional steel	31,243.37	45,699.20	76,942.57	105,712	0.73

using costs based on actual operations from January to November, 1951. Table V gives a summary including all stope development, with the exception of the scam drift, and all mining costs. It is assumed in each case that all ore is handled from the stope to the transfer raise through a scam drift.

It is felt that as the miners become more accustomed to the air-leg drills a greater reduction in

shrinkage costs can be effected. Sectional steel has been in use since June, 1951, and accounts for a footage approximately one fifth of that drilled by diamond drills. Each month has seen a reduction in cost per ft of hole drilled with sectional steel, so there is justification in predicting further reduction of sub-level mining costs by its use. The slightly larger hole resulting from the sectional steel drilling as compared to diamond drilling permits a heavier powder concentration per ft of hole; consequently in current drilling practice the interval between rings is being increased from 7 to 7½ ft, a move which should reduce further the overall cost per ton.

Cut-and-Fill Mining

Horizontal cut-and-fill stoping as yet accounts for only a small percentage of the total tonnage mined or broken. This method is confined at present to one orebody in the hanging wall between the 300-ft level and surface. Eight-foot cuts are taken with stopers, holes being drilled 10½ ft deep at 50° and blasted with short period delay blasting caps. Better fragmentation and drilling cycle through this practice, see Table VI, has improved the overall performance above that obtained with horizontal holes and standard delay caps or safety fuse.

Table VI. Comparative Performance Data in Cut-and-Fill Stopes

Method of Drilling	Tons Broken Per Man Shift	Powder Per Ton, Lb	Tons Slushed Per Man Shift	Powder Per Ton, Lb, Sec. Break
Flat holes, drifter	32.8	0.46	29.6	0.07
Inclined holes, stoper	35.6	0.50	65.8	0.03

Ore is scraped from the stopes and the fill is spread with a double-drum 15-hp air hoist and a 48-in. scraper. Chutes are built of 8x8-in. cribbing and lined with round lagging flattened on one side. Round lagging at 10 to 12 in. centers is used for flooring and waste rock from the mine is used for fill. It is anticipated that cut-and-fill mining will be used to a much greater extent when greater depths are reached. To this end considerable re-

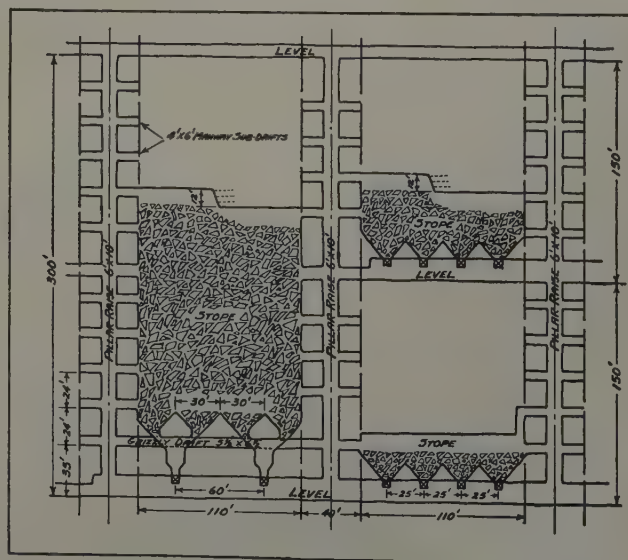


Fig. 9—High sill shrinkage stope, left, and low sill shrinkage stope, right. Orebodies under 30-ft width are developed at 150-ft vertical intervals, wider orebodies at 300-ft intervals.

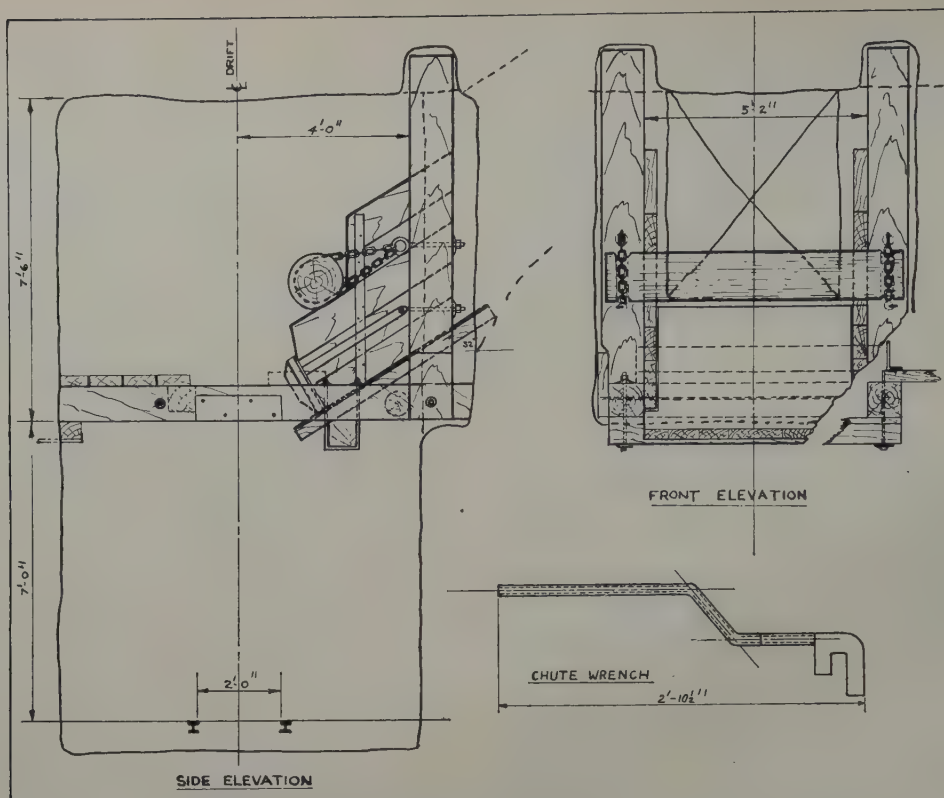


Fig. 11—Standard timbered boxhole chute has sides and bottom prefabricated in shop.

The collar of this raise on the haulage level below is equipped with an efficient steel and concrete chute with an air-operated undercut arc gate, see Fig. 12. The chute has a throat opening 5 ft wide by 4 ft normal to the flow of the ore. At the top of this raise finger raises are driven to serve as many as four drawpoints in adjacent scam drifts. The tops of the finger raises are protected by a discarded cone crusher liner having an opening $3\frac{1}{2}$ ft in diam, set

Table VIII. Comparative Efficiency and Direct Labor Cost Of Ore Haulage in Different Types and Sizes of Cars

Haulage	Tons Per Man Shift		Direct Labor Cost Per Ton	
	90 Cu Ft	110 Cu Ft	90 Cu Ft, \$	110 Cu Ft, \$
Low sill	50	57	0.20	0.18
High sill	97	131	0.105	0.075

in concrete. General practice is to locate the scam drifts under adjacent stopes along the strike of the orebody so that one permanently located slusher hoist will handle the broken ore from two stopes by simply reversing the direction of the cables.

From the scam drifts, boxholes 6 ft wide and $4\frac{1}{2}$ ft high are cut at approximately 30-ft centers and driven at 45° to the sill elevation. They are then coned out and the center of the brow is opened up a bit to form an arch. Two-drum, 60-hp electric hoists having a rope pull of 8150 lb at 240 ft per min are used for slushing. Discarded hoisting ropes $\frac{7}{8}$ to 1 in. in diam are used and wire rope cost, per ton scammed, averages 6 to 10¢. The scrapers used are 66-in. hoe type and weigh 2735 lb. As stated above, the slushers under sub-level stopes are located permanently on a base, but under shrinkage stopes slusher hoists mounted on trucks are used.

This permits maximum use of equipment, since when the drawing cycle is complete in one shrinkage stope the slusher can be moved to another ready for pulling. Slusher performance under shrinkage stopes averages 297 tons per scraper shift and under sub-level stopes 274 tons per scraper shift. Advantages of the use of scam drifts are relatively high rate of production, less pillar requirement under a stope, and higher production per man shift.

In Table VII the three methods of handling broken ore from the stope to the ore pass are compared on a cost per ton and production basis.

Included in the above scamming costs are maintenance costs of 2¢ for scrapers and 8¢ for concreting scam drifts. This latter figure is expected to be reduced by the present practice of locating the scam drift in the foot wall of the ore zone.

From the above table it is evident that scamming production per man shift is higher, a very important item in the present tight labor market. Rate of production from a stope with scamming facilities is 36 pct higher than in the high sill method and 169 pct higher than from a low sill stope.

Haulage Equipment

Level haulage underground is done entirely by battery locomotives. On development work 2-ton motors are in general use and for production, 4-ton and 6-ton locomotives. The 4-ton locomotive complete with battery weighs 12,000 lb and the 6-ton complete with battery weighs 20,000 lb.

Use of battery locomotives exclusively for underground haulage was decided for the following reasons: 1—They allow complete flexibility of movement of equipment on any level. 2—Although tonnage from any level is high, it originates from several sources on a level, and thus equipment to handle it is always available. 3—Ore passes are well

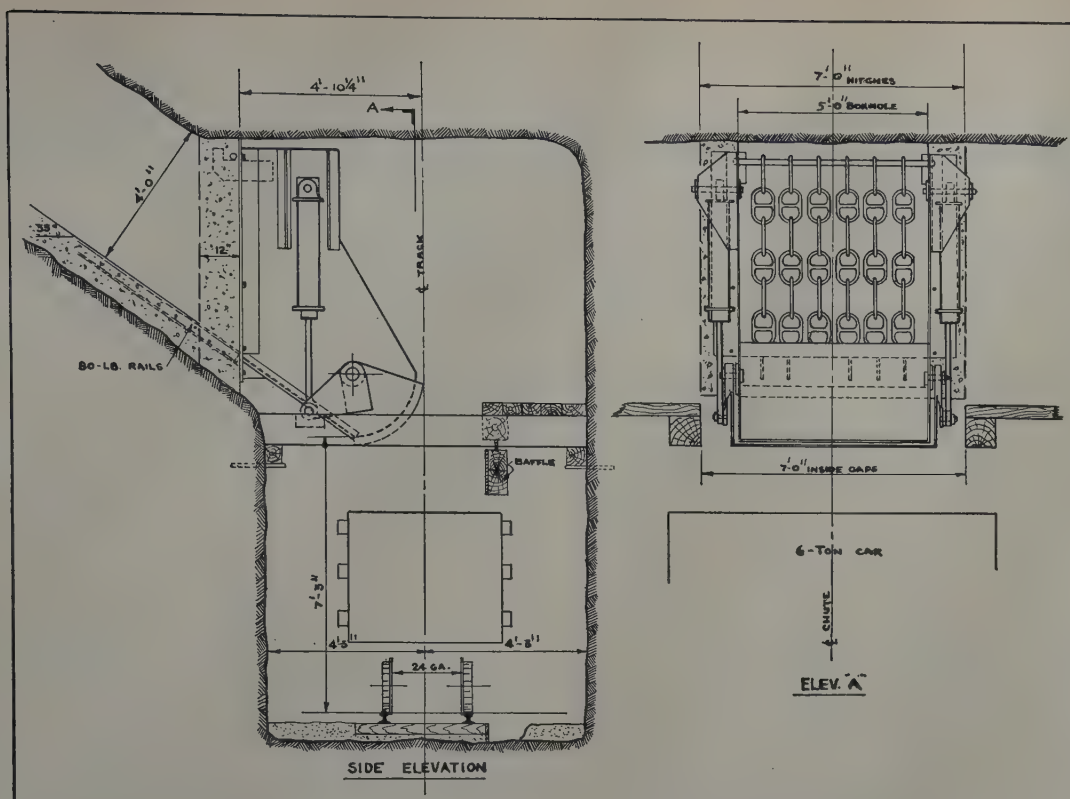


Fig. 12—Air-operated undercut arc gate is used on standard loading chute No. 3.

located so that average tramming distances are relatively short. In fact, for the first ten months of 1951 distance trammed averaged only 1260 ft. 4—There is relative safety because of low voltages used in battery as compared to trolley locomotives. 5—There is low fire hazard, as all electrical equipment is localized in level charging station with battery locomotives.

Cars used for development work are 2-ton Hudson-type and 65-cu ft Granby-type cars. For ore haulage the original cars used were 90-cu ft Bell-type design. The bottom of this car slopes at 40° toward a side opening door. The door is locked in the closed position and is released and opened at the ore pass by a wheel passing over a ramp. The car

For these reasons 110-cu ft Granby-type cars were purchased. The size of the service cage governed the size of the car, and the ones purchased can be loaded on the cage after the ramp wheel and couplers are removed. The car is 11 ft 12 in. long overall, 4 ft 6 in. wide, and weighs 10,500 lb. The door is hinged to swing freely to allow dumping the large pieces of ore without barring. The undercarriage of the car is cast steel and adjacent to each wheel are located a leaf spring and two helical springs. The latter are located directly over the rail and, since these cars are operated on 24-in. gage track, have an essential stabilizing effect on the car. As expected, the car factor for this car was 24 pct better than for the Bell-type 90-cu ft car.

As the 4-ton locomotives handle only five cars of either the 90-cu ft Bell or 110-cu ft Granby type, comparative performance figures for each kind of train on low and high sill haulage are available. Table VIII gives this information for the first nine months of 1951.

Thus in low sill haulage tons per man shift increase 14 pct and direct labor costs decrease 10 pct. In high sill haulage where tram equipment is used to better advantage an increase of 35 pct in tons per man shift and a decrease of 28.6 pct in direct labor costs were brought about.

By the purchase and use of the 6-ton locomotives further economies were effected, since fifteen 110-cu ft cars could be drawn in a train as compared to five drawn by the 4-ton locomotives. Table IX shows the comparative performance of the 4 and 6-ton locomotives.

Use of the larger locomotives resulted in increases in tons per man shift in low sill haulage 16 pct, high sill haulage 41.5 pct, and from transfer raises 51.7 pct. Similarly, direct labor costs per ton hauled were decreased 13.9 pct, 26.7 pct, and 31 pct respectively for the three types of haulage.

Table IX. Comparative Performance of 4 and 6 Ton Locomotives in Three Types of Haulage

Haulage From:	Tons Per Man Shift		Direct Labor Cost Per Ton, \$	
	4-Ton Loco.	6-Ton Loco.	4-Ton Loco.	6-Ton Loco.
Low sill	57	66	\$0.18	\$0.155
High sill	131	185	0.075	0.055
Transfer raises	153	232	0.065	0.045

chassis is mounted on two 4-wheel trucks, a feature allowing the car to be turned at any switch. This type of car was chosen in anticipation of a 1000-ton operation because the sloped bottom would allow cleaner dumping of the ore, the fines of which were sticky. However, in more recent years, in the handling of coarser ore, much time has been lost at the ore pass because it has frequently been necessary to bar large pieces out of the cars. Moreover, the design of the car renders it topheavy when loaded. Also the plan area of the car is greatest at the top, where of course the effect of variation in loading is at its maximum.

Operating Behavior of Liquid-Solid Cyclones

by E. B. Fitch and E. C. Johnson

The operating behavior of liquid-solid cyclones is outlined, together with the nature and range of the process results obtainable, to serve as a background for engineers wishing to consider application of this new process tool.

BY now most engineers are familiar with the liquid-solid or Dutch State Mines cyclone. However, it should be helpful to know exactly what it is that the equipment does and what its limits are. Without going into cyclone theory, this paper will describe the operating characteristics of Dutch State Mines cyclones. These are manufactured under license in this country and sold under the trademarked name of DorrClone. The physical construction of the liquid-solid cyclone has been covered in many papers,¹⁻⁴ the DorrClone in particular being described in some detail by Weems.⁵

Fig. 1 shows the unit in cross-section. The feed enters at C. The coarse, heavy particles are thrown centrifugally to the periphery and make their way down the wall to the apex where their rate of discharge as underflow is controlled by an adjustable rubber apex valve. As the apex diameter is decreased the solids build up behind the valve, producing a denser underflow. Meanwhile the fine particles are swept into the upward flowing vortex stream which exits as overflow through the vortex finder, F.

Flexibility to produce the specific result desired in a particular process is achieved by providing means for varying the areas of the entrance, vortex discharge, and apex discharge. The entrance area may be varied by insertion of special shims. Vortex discharge area may be changed by use of different-sized vortex finders which are interchangeable. Similarly, the different sizes of apex valves are interchangeable and in addition each apex valve is variable down to about 60 pct of its maximum diameter.

A most significant primary distinction to make is that although liquid-solid cyclones have been sometimes called thickeners, they actually are classifiers, and very potent ones. They are almost never thickeners in the special sense that many metallurgical engineers understand the term. There would be no profit in quibbling over the definition of a word, but when the application of cyclones is considered, it will help to understand the difference be-

tween two mechanisms, one of which will be called *classification*, and the other *thickening*.

In what is called thickening the fine solid particles present in the feed hold together by surface attraction during the sedimentation process. The loose network of particles thus held together constrains all particles to settle at approximately the same rate, the larger ones dragging the smaller ones down. As a result, pulp settles with a sharp line of demarcation between solids and a relatively clear supernatant liquid. Essentially all the solids, regardless of their fineness, pass into the thickened underflow, and a clear overflow is separated.

In classification, on the other hand, the interparticle forces are relatively insignificant as compared to the settling force on the individual particles, and are insufficient to prevent independent movement of the particles. The coarsest, heaviest particles settle most rapidly through the pulp, passing more slowly settling fines. Particles coarser than the mesh-of-separation essentially all settle into the underflow, but if the feed contains any particles finer than the mesh of separation, at least part of them will appear in the overflow. A clear supernatant or overflow can be obtained only if there are no undersize particles present in the feed.

Thus it will be seen that classification is impossible under ideal thickening conditions. The finer particles are pulled down at essentially the same rate as coarser particles, and there is no separation on the basis of particle size.

The surface attraction holding the particles together in a thickener is usually feeble. Whenever the sedimentation force on any particle is strong with respect to the interparticle forces, that particle can pass through the tenuous structure and settle independently. There are at least four ways of making the sedimentation force strong, with respect to the interparticle forces, and obtaining classification. First, and most obvious, the particles may be large and heavy. Thus coarse sands settle out in a beaker or Dorr thickener ahead of the rest of the thickening solids. Second, the interparticle forces may be altered by physicochemical means; i.e., it is often necessary to add dispersing agents to destroy the interparticle forces and permit classification to take place. Third, the interparticle forces may be reduced by dilution of pulp. It is well known that to obtain the most efficient separation of

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Discussion on this paper, TP 3445B, may be sent (2 copies) to AIME before May 31, 1953. Manuscript, Feb. 20, 1952. New York Meeting, February 1952.

finer and oversize in a gravity-sedimentation or Dorr-type classifier it is necessary to dilute the feed to some appropriate value. Fourth, the sedimentation force may be increased through substitution of a centrifugal field of many G's for the simple force of gravity. This happens in a cyclone, and partly explains why a cyclone nearly always operates as a classifier and not as a thickener.

There is another and perhaps even more important mechanism which acts to make the cyclone a potent classifier rather than a thickener. As has been described by several writers,^{1,2,4} the fluid in a cyclone does not move at constant angular velocity as it tends to do in a centrifuge, but speeds up as it moves towards the axis. The velocity gradient sets up shear forces in the fluid which overcome the natural flocculence of any but most the strongly flocculated pulps. Fine particles are prevented from attaching to larger particles and settle separately.

From the above, it will be seen that a cyclone essentially always acts as a classifier. It classifies small particles, and it can do so without the expedients of added dilution or dispersing agents. This characteristic is one of the most valuable special assets of the cyclone.

Mesh of Separation

The first question to be asked about a classifier is what separation it makes. Here there are two difficulties. To begin with, there is some confusion in terminology. In literature on cyclones the separation has frequently been designated in terms of the size particle which reports half to the underflow and half to the overflow. In metallurgical practice it is perhaps more common to designate separation in terms of the largest sized particles that appear in the overflow in any significant quantity, this being commonly termed the *mesh of separation*. In classification by ordinary gravity sedimentation, it makes little difference which point is described, as long as it is known which one is intended. In this case the 50 pct point is generally a calculable amount, in the range of the second screen partial, finer than the mesh of separation. In cyclones, as will be shown later, there is no such predictable relationship. This paper, therefore, will refer always to the mesh of separation, according to common metallurgical practice, rather than to the 50 pct point referred to in the literature on cyclones.

Table I. Separations Obtained with Various Fittings in a 3-in. DorrCone Separator Operating on a Dilute Suspension of -65 Mesh Dolomite Solids in Water, Fed into the Cyclone at 20 Psi

Feed Entrance, Area in Sq In.	Vortex Finder, Diam, In.	Separation, Microns
0.75	1.05	34
0.53	0.82	28
0.30	0.62	24
0.19	0.49	23
0.10	0.36	27

In the second place, there are a surprisingly large number of variables significantly affecting the separation made by this mechanically simple piece of apparatus. It could be misleading to give the range of separation to be expected from the different sizes of units without explaining how this may vary, and without stating fully the conditions for which the specified range would be valid. For this reason the effect of the more important variables on the mesh of separation will be discussed, and data will be

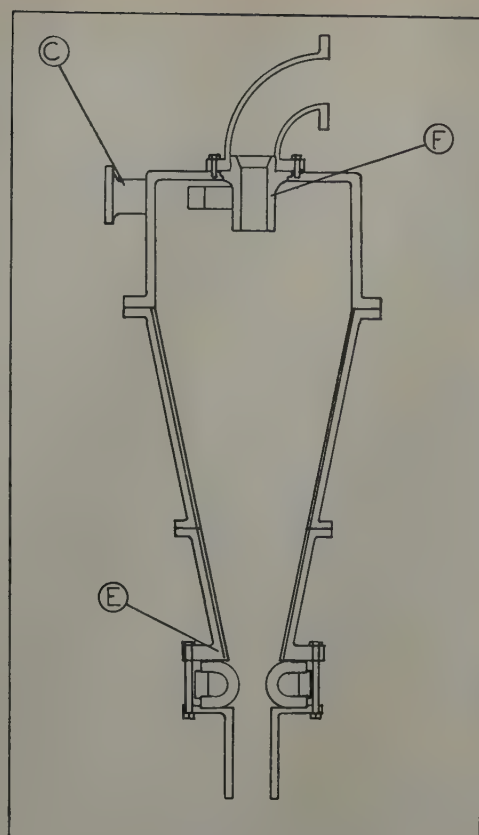


Fig. 1—DorrCone with tire-type apex.

given on the range of separation of the different sizes of DorrCone units under specified conditions.

Given a series of cyclones geometrically similar except for the sizes of the feed, and the overflow and underflow orifices, the following have been found significant variables in determining the mesh of separation produced: 1—Size of feed entrance and vortex finder, 2—Size of cyclone, 3—Specific gravity of feed solids, 4—Quantity and size distribution of feed solids, 5—Pressure drop or head loss across cyclone, 6—Plasticity of feed pulp.

Size of Feed Entrance and Vortex Finder

The prime adjustment for mesh of separation is the size of the feed entrance and vortex finder. Larger fittings give coarser separations. Smaller fittings, down to a certain critical size, give finer separations. However, for any given size of cyclone, making fittings any smaller than this critical size results again in coarser separations, rather than in finer ones. For minimum mesh of separation in any given size of cyclone, the feed entrance and the vortex finder usually have equivalent diameters in the range of 1/5 to 1/6 the diam of the cyclone. However, this has been observed to vary with the nature of the feed slurry.

To illustrate the effect of changing the feed entrance and vortex finder, Table I shows the separations obtained in a 3-in. cyclone operating on a dilute slurry of -65 mesh dolomite solids. The operating pressure was constant at 20 psi. In every case the feed entrance has an area essentially equal to that of the vortex finder. Each point is the average value for seven to nine individual test runs.

Smaller cyclones can make finer separations than larger ones; they cannot be fitted to give separations as coarse. This will be apparent from the data on the range of separations possible with the different

sizes of cyclones. Thus the minimum separation obtained with a 12-in. cyclone under the conditions as specified for Table I was 37 microns, as compared to the minimum of 23 microns shown in Table I for the 3-in. unit.

Table II. The Effect of Dilution on Separation

Dilution in Cyclone Overflow, Volume	Solids in Cyclone Overflow, Wt Pct	Separation
4	42	coarser than 147 mu, 100 mesh
5	37	60 micron
6	32	43 micron, 325 mesh
10	22	36 micron
20	13	32 micron
Very high	Low	29 micron

It is obvious that the specific gravity of the solids affects the separation size. Dahlstrom's correlation² showing separation inversely proportional to $(\rho_s - \rho)^{1/2}$ has proved satisfactory where ρ_s = sp gr solid and ρ = sp gr slurry.

The mesh of separation obtained in a cyclone is strongly affected not only by the quantity of solids in the slurry treated, but also by the size distribution of the solids. As the solids become crowded in a slurry the particles interfere with one another and settle more slowly. The mesh of separation becomes coarser. Increasing coarseness of the solids also coarsens the mesh of separation.

The effect of dilution is illustrated qualitatively in Table II. In this case the feed was a 95 pct -325 mesh dolomite slurry of varying dilution. A 12-in. cyclone was used. Separation is here correlated with overflow dilution, although it is not established that this is the best correlation to use. In Table II volume dilution is defined as the volume of liquid in a slurry divided by the volume of solids.

While it is not safe to draw any quantitative conclusion from Table II, it is clear that as solids become crowded in a slurry, separation becomes coarser.

Experience shows that the effect of dilution can usually be disregarded for purely practical purposes until the volume dilution of the feed drops below 8. At some dilution less than this it begins to be very important. At just what dilution the rapid increase in diameter-of-separation sets in seems to depend upon the size of the unit, the fittings used, and probably upon the coarseness of the solids. Table III

Table III. Percent Solids at Volume Dilution of 8

Solids, Sp Gr	Solids, Pct
1.3	14
2.0	20
2.7	25
3.0	27
4.0	33
5.0	39

shows the weight percent solids at a volume dilution of 8 for solids of different specific gravities. If the percent of solids in a slurry is much higher than shown in Table III, separations substantially coarser than those to be mentioned below must be anticipated.

The effect of coarseness of feed can be seen from the following examples. Under essentially identical operating conditions, a 3-in. DorrClone unit gave a 23-micron separation when treating -65 mesh dolomite solids, and a 16-micron separation on -325 mesh solids. A 12-in. DorrClone unit gave a 37-micron separation on -65 mesh dolomite solids, and

a 29-micron separation on -325 mesh solids. Even though the difference in feed did not appear to be great, there was a substantial difference in the separation. It will be apparent, therefore, that separations in a cyclone may be made substantially finer if the feed is first scalped of coarse particles. The scalping device may be another cyclone.

Increasing pressure on a given cyclone results in finer separations. As a generality the diameter-of-separation seems to vary inversely as something less than the fourth root of pressure; thus doubling the pressure would give as a maximum about a 16 pct decrease in the diameter-of-separation. At the same time the slurry throughput would increase about 40 pct.

Direct data available at this time are not consistent enough to permit accurate evaluation. Practically, separation is made finer by increasing pressure, but only big changes in pressure make a noticeable difference. An exception may exist in the treatment of plastic pulps.

Plasticity of Feed

If the pulp solids contain more than 2 or 3 pct undispersed clay or claylike solids, the pulp will usually be significantly plastic. In general, separations on plastic pulps will be from one to many meshes coarser than would be obtained on nonplastic slurries. Increasing pressure is particularly beneficial in treating plastic pulps, because the increasing centrifugal forces thus obtained aid in causing particles to tear their way through the plastic structure of the solids. Therefore the mesh-of-separation may become finer more rapidly than normal with plastic pulps, and at very high pressures, as in the range of 70 psi, the separation may often approach that expected for nonplastic slurries. A plastic pulp may usually be recognized by the paintlike coat of solids covering objects dipped in it.

The separation ranges given in Table IV are valid for the feed condition which comprises dilute suspension of -65 mesh nonplastic solids, fed through the cyclones at a head of 50 ft. The ranges are covered by changing the feed entrance areas and the vortex finder diameters. The separation will deviate from the given ranges with changing conditions as described above.

Sharpness of Classification

Mesh-of-separation does not tell the whole story on classification. The next question is how sharp a classification is produced. How free from fines is the coarse product or underflow?

In considering this question it will be helpful to analyze the mechanism of classification by sedimentation. This is shown most clearly in batch sedimentation. Consider a beaker of slurry, containing solids sufficiently dispersed so that classification and not thickening takes place. Originally, as a result of stirring, all particle sizes are uniformly distributed throughout the slurry. As soon as agitation stops, all particles start settling towards the bottom of the beaker at a rate determined by their size and density, among other factors. Those reaching the bottom settle out. After some given time, the slurry above the deposited solids is poured off or otherwise removed, leaving the deposited solids behind. The deposited solids, together with entrained water, represent the underflow of a classifier. The slurry removed represents the overflow.

There is a certain size of particle which during the sedimentation has just time to settle from the top

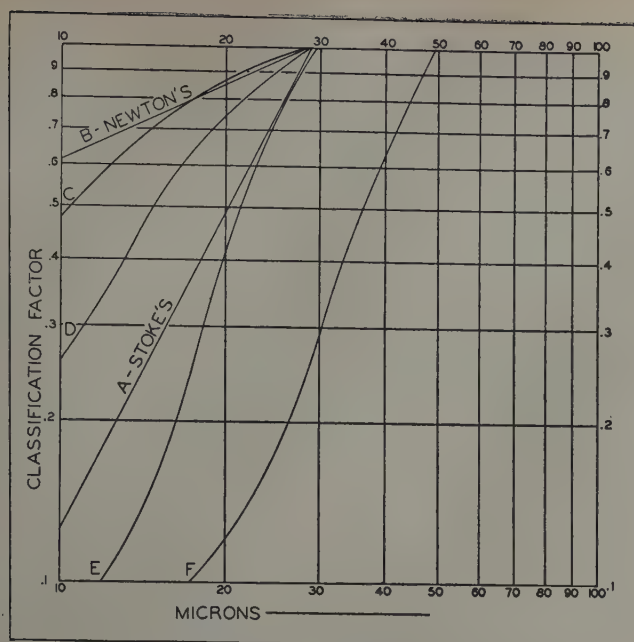


Fig. 2—Classification factor vs micron size.

surface of slurry and deposit on the bottom of the beaker. All particles coarser than this will have already settled out and thus will be totally removed from the overflow fraction. Smaller particles will not reach the bottom or underflow fraction if they start from the top of the pulp, but a fraction of them starting lower in the beaker will reach the bottom in the time allowed. In other words, all particles coarser than the mesh of separation will be collected in the settled fraction and will be absent from the residual overflow suspension. Some fraction of any size range of particles finer than the mesh of separation will also settle into the underflow, the relative quantity depending upon their settling rate.

In addition to particles which travelled into the underflow fraction by settling, there will also be a part of the feed slurry originally occupying the space at the bottom of the beaker which will be invaded later by the settling solids. Some of this is displaced by the settling solids, but some also remains to fill the void spaces between the settled solids. This void filling carries into the underflow an appreciable amount of slimes, or particles so fine as to have insignificant settling rates.

Table IV. Flow and Separation Ranges Expected in Standard Dorr-Clone Units When Fed Dilute, Nonplastic Slurries Comprising —65 Mesh Solids of Various Specific Gravities Suspended in Water, Operating at a Head of 50 Ft, With Various Feed Entrances and Vortex Finders

DorrClone Unit, Diam, In.	Feed Flow, GPM	Separation, 1.35 Sp Gr Solids, Microns	Separation, 2.7 Sp Gr Solids, Microns	Separation, 4.0 Sp Gr Solids, Microns
3	8 to 46	50 to 85	23 to 40	17 to 30
6	43 to 152	62 to 100	28 to 45	21 to 33
12	160 to 550	82 to 128	37 to 58	28 to 44
24	465 to 1460	115 to 150	52 to 67	40 to 50

The underflow from a simple sedimentation classifier, then, comprises three different classes of particles: those coarser than the mesh of separation, or oversize; those having appreciable settling rates but finer than the mesh of separation, or critical sizes; and void filling, which mathematically may be considered to have the same composition as overflow

slurry. Although the mechanism has been discussed in terms of simple batch sedimentation, calculations based on its performance have been found valid for continuous sedimentation also.

In practice it has been discovered that the overflow, instead of showing 0 pct plus the mesh of separation, in a screen analysis will show from 1 to 2 pct plus the mesh. Because of shape factors particles are sorted differently by a screen than by a sedimentation process. That is, the screen treats a flaky particle, such as a piece of mica in the extreme case, as much larger than does a sedimentation pool. If the particles were all of exactly the same gravity, and were geometrically similar in shape, then there should be no discrepancy. The shape factors of ordinary milled ores are such as to give about 1½ pct plus the mesh of separation in the overflow. Thus sharpness of classification depends upon the amount of overflow slurry pulled into the underflow as void filling or entrainment, and upon the distribution of critical sizes between the overflow and underflow.

It will be seen that the amount of void filling, and hence the fraction of the feed slimes which appear in the underflow, is determined by the split of water between the two fractions. The less water in the underflow, or to put it another way, the higher the percent solids in the underflow fraction, the more complete the desliming.

In desliming effectiveness, the cyclone can approach an ordinary rake-discharged bowl classifier. It can be substantially more effective than an Hydroseparator. Under comparable conditions, a rake-discharged classifier might give from 22 to 28 pct water in the underflow, a cyclone from 27 to 30 pct water, and a Hydroseparator about 50 pct water. Therefore, a cyclone, if operated with as thick an underflow as feasible, will have in its underflow about the same percent of slimes as a rake-discharged classifier, but only about 60 pct as much as obtained from a Hydroseparator. It can be seen that if desliming is desired, the underflow from a cyclone, or any simple sedimentation classifier, must be run as thick as possible.

In ordinary gravity sedimentation, little control can be exercised over the disposition of the critical sizes. In a cyclone, on the other hand, the split of critical sizes between the underflow and the overflow is strongly affected by operating conditions. A cyclone can be operated to give a separation much sharper than obtained by gravity sedimentation, but it can also be operated to give separations much less sharp.

The following factors have been found to have significant effect upon the distribution of the critical sizes and hence upon the sharpness of classification: percent solids in underflow, size of feed entrance and vortex finder, size of cyclone, and pressure.

The percent solids in the underflow is controlled by the setting of the apex valve, which in turn controls the flow ratio, or gpm underflow divided by gpm feed. A high percentage of solids in the underflow tends to increase the fraction of critical sizes thrown into the overflow. At the same time, as described above, the high percentage of solids gives decreased void filling. Both factors combine to increase sharpness of classification and decrease recovery of feed solids in the underflow. This gives wide variations in the 50 pct separation point but has been found to have relatively little effect on the mesh of separation.

Fig. 2 shows classification factor curves observed for a 3-in. DorrClone unit operated under conditions which were essentially constant except for the underflow percent solids and rate.

The classification factor curves of Fig. 2 are log log plots of equivalent diameter of particles, against the fraction of the particles of that equivalent diameter which settle out during sedimentation as described above. Curve A in Fig. 2 shows what the factor would be if the particles all settled according to Stokes' law. Curve B shows what they would be if the particles all settled according to Newton's law. Note that they are both straight lines on a log log plot. Classification by gravity sedimentation, particularly at fine meshes of separation, will show factors not greatly different from Curve A. Furthermore, the factors will not change significantly with changing underflow rates and percent solids.

Curves C through F show the factors observed for different flow ratios, and hence for different percent solids in the underflow, under conditions shown in Table V.

Note that the curves of Fig. 2 were calculated from runs made with a 3-in. DorrClone unit. In the larger units, the effect is much less pronounced, and the factor curves tend to resemble E of Fig. 2, rather than C, over a much wider range of underflow dilutions. Thus the factor curves, hence also sharpness of classification and recovery of solids in the underflow, are much less influenced by underflow dilution in larger cyclones than in smaller ones.

In all sizes of cyclones, decreasing the size of the feed entrances and vortex finders gives poorer classification and higher recovery of solids to the underflow. Increasing pressure appears to have the same effect.

Table V. Conditions of Tests Showing Effect of Underflow Percent Solids on Classification Factor

Item	All Runs			
DorrClone unit, diam	3 in.			
Feed entrance	0.19 sq in.			
Vortex finder	0.49 in.			
Pressure	37 psi			
Solids in feed slurry	95 pct — 325 mesh dolomite			
Solids in feed, pct	43 pct			
Runs	C	D	E	F
Solids in overflow, pct	17.7	21.9	26.0	32.4
Solids in underflow, pct	51.8	57.4	62.0	71.1
Underflow gpm divided by feed gpm	.79	.51	.38	.19

The curves of Fig. 2 represent the extreme case. They were calculated from runs on the smallest of the DorrClone units, operating with a small feed entrance and vortex finder, and at a relatively high pressure. The effect will be far less with larger units, and is not particularly noticeable in a 12-in. machine.

Curve F of Fig. 2 is interesting. In all other runs the underflow was discharging in a spray. When there is a spray discharge, the apex orifice is not running full of slurry, but has an air core at the center. As long as a spray discharge is obtained, it is believed that the apex orifice is not overloaded, and no underflow material is backing up in the cyclone to discharge out the overflow. As the apex valve is constricted further and further, a point is reached at which the character of the underflow discharge changes abruptly. The underflow starts flowing in a steady, ropelike stream rather than spraying, and the air core disappears at the apex.

This may represent an overloaded condition, wherein underflow is not allowed to escape at the apex as fast as it collects. Curve F shows the results. The mesh of separation becomes coarser. Also a substantial amount of stray or short-circuiting overflow may appear in the overflow.

To obtain a sharp classification, then, the apex valve should be adjusted to give as dense an underflow as possible without rope discharge. This is particularly important when making fine separations, which call for small diameter units, with small fittings and high operating pressures.

Sharp classification is not always desired. Although it has been shown that cyclones are classifiers rather than thickeners, they have been used as substitutes for thickeners where fines can be tolerated in the overflow. In fact, this was one of the original uses for which they were developed by the Dutch State Mines. When substituting for a thickener, it is desirable to collect just as many of the solids as possible in the underflow. Under such conditions the cyclone would be operated to be a poor classifier, throwing critical sizes as much as possible into the underflow rather than into the overflow. Cyclones for this job will normally be as small in diameter as is consistent with operating convenience and will have relatively small ports. They will be operated at high pressures, with as dilute underflows as possible processwise.

Operating Requirements

Interchangeable feed entrance shims and vortex finders give a DorrClone unit considerable flexibility. Without this feature it would not be feasible to adjust cyclone behavior for optimum process performance, or to make the changes in mesh of separation dictated from time to time by process considerations. However, once the unit is set up with the selected fittings, it is essentially a constant flow machine. Means must be provided to maintain the feed flow required by the machine under the chosen conditions. Suitable means have been discussed by Weems.⁵ They comprise surge tanks, controlled additions of water where permissible, and recirculation of machine overflow.

Control of the apex valve is often necessary. Small fluctuations in the amount of underflow solids to be discharged will not greatly change the percent solids in the underflow, because in this respect cyclones have a limited self-regulating tendency. To accommodate large changes, if a constant and high percentage of solids in the underflow or a sharp classification is desired, it is necessary to control the apex valve diameter. Where the changes in feed slurry composition are gradual, manual control of the apex valve will often suffice. To handle rapidly changing conditions, an automatic apex control system⁵ is available.

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Sampling and Testing of Sinter

by R. L. Stephenson and D. J. Carney

A sampling technique has been developed for procuring a sample of sinter representative of the entire depth of the sintering bed. The sampling method involves the use of an open-bottom metal basket that rides on the grate of the sintering machine and when removed contains a sample of the sintered product. Additional data have been obtained to indicate that the tumbler test is a suitable means of measuring sinter strength.

IN the last few years additional sintering facilities have been installed in both the Pittsburgh and the Chicago district of the United States Steel Co. Since the construction of these sintering plants made possible the use of higher percentages of flue-dust sinter in our blast-furnace burdens, it became important to study means of controlling the quality of sinter to obtain optimum results in the blast furnace.

For controlling an operating process, it is necessary first to establish standards by which the quality of the product can be judged. For sinter, it appeared that an important property was its strength or its resistance to degradation during transportation and charging into the furnace. Consequently work was undertaken to establish a standard for sinter strength that could be used both for controlling sintering-plant operations and for correlating sinter quality with blast-furnace performance.

The first problem in setting up a standard was that of procuring a sample that would be representative of the sinter made under any particular set of conditions at the sintering plant. Since the United States Steel Co. sintering plants discharge the finished sinter either into a large pit or onto a rotary cooler, the sinter becomes inseparably mixed with material sintered 2 hr before or 2 hr afterwards. For this reason the exact identity of the sinter is lost. A sample selected as the cooler is discharged, or as the sinter is removed from the pit, cannot be said to be truly representative of the sinter made at any specific time.

Sampling

The first attempt to procure a sample that would be representative of a specific sinter mix and of specific operating conditions was made by stopping the Dwight Lloyd sintering machine and removing an entire pallet full of sinter. This method, however, proved very difficult to perform and interfered considerably with the operation of the plant. To overcome this difficulty, a sampling method was devised by technologists at South Works enabling them to secure, without interrupting the sintering operation, a sample of about 1 cu ft of sinter, representative of sinter for the full depth of the sintering bed.

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Discussion on this paper, TP 3508B, may be sent (2 copies) to AIME before May 31, 1953. Manuscript, Oct. 21, 1952. Chicago Meeting, September 1952.

The South Works method involves the use of a steel-frame-work basket. A typical basket is shown in Fig. 1. The basket has been used both with and without crossbars along the bottom. As long as the crossbars are in the same direction as the grate bars on the sintering machine they do not interfere with the sintering process. The basket is set on an empty grate of the Dwight Lloyd sintering machine before it passes under the swinging feed spout, see Fig. 2. When the basket is removed after it has travelled the length of the sintering machine, it contains the sample. Just before the basket is removed, the sinter is scored and chipped to facilitate removal of the sample from the sinter bed. A view of the basket after its removal is shown in Fig. 3. Although the sampling method was originally designed for use on a Dwight Lloyd sintering machine, it can also be used on the Greenawalt type of machine. When used on the Greenawalt-type machine, the basket is placed on the sintering grate before the charging car passes over it, and finally it is removed just before the pan is dumped.

Testing

After a method of obtaining a representative sample of sinter had been developed, the next step was to select a method of measuring its strength. The irregular shape and size of the sinter pieces precluded the use of a simple compression test for determining strength; consequently, the shatter test and tumbler test were investigated. To perform the shatter test, a sample of sinter, approximately 5 lb, is dropped from a hinged-bottom box at a height of 3 ft onto a steel plate. The broken sinter is sieve-analyzed after a specified number of drops. The tumbler test is performed with the use of a standard ASTM coke-tumbling drum. The drum is 3 ft in diam and is equipped with two lifter bars diametrically opposite one another on the inner periphery of the drum. The drum is rotated at a speed of 24 rpm for 200 revolutions, and after tumbling the sample is sieve-analyzed.

To express as single numbers the results of sieve analyses after shattering or tumbling, the method suggested by R. E. Powers¹ was employed. This method involved plotting the size of the sieve openings on a logarithmic scale and the cumulative percent larger than each sieve on a probability scale as described by J. B. Austin.² By interpolating from the plotted data, which in most cases approximated



Fig. 1—Sampling basket, United States Steel Co.

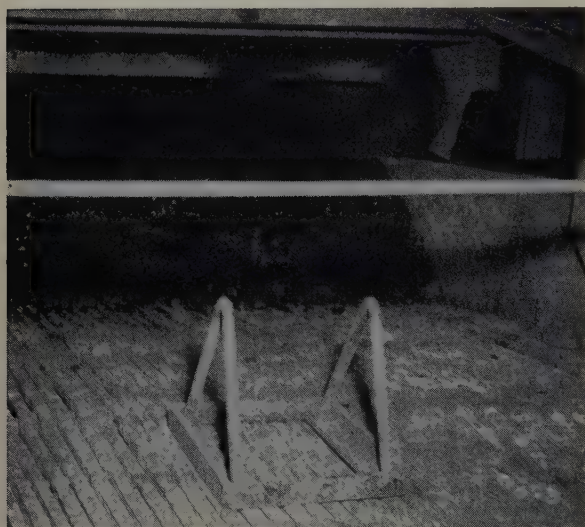


Fig. 2—Sampling basket in place on an empty sintering grate before passing under the swinging feed spout.



Fig. 3—Sampling basket, containing sinter sample, after removal from the sintering bed.

a straight line, the median size could be determined. The median size is taken as the point of intersection of the curve with the 50 pct ordinate.

Results of shatter tests indicated that this test was not severe enough, in that the median size of the sinter after shattering was greatly affected by the starting size of the original sample. On the other hand, the results of tumbler tests revealed that the median size of the sinter after tumbling was almost unaffected by the starting size, and that this test was still selective enough to distinguish clearly between median sizes of sinters of different strength.

Fig. 4 illustrates the effect of starting size on the distribution of particle sizes after shatter testing for 16 drops. From this plot it can be observed that the median size is affected by the size of the pieces selected for performing the test. With a starting size of between $\frac{1}{2}$ and $\frac{3}{4}$ in., the median size after testing was 0.398 in., whereas with a starting size of between 1 and 2 in., the median size after testing was 0.762 in. Fig. 5 illustrates the effect of starting size on tumbler-test results. In this plot it can be seen that the starting size had little effect on the median size of the sinter after tumbling, for when the starting size was $\frac{1}{2}$ to $\frac{3}{4}$ in. and when it was 1 to 2 in. the resulting median size was 0.163 in.

Although the shatter test is not severe enough to eliminate wide discrepancies in median size caused by variations in starting size, the tumbler test is severe enough to reduce such discrepancies to a minimum. Furthermore, the tumbler test is not so severe that it destroys the distinction between sinters of different strength. This fact is illustrated in Fig. 6, which shows a plot of tumbler-test results for a sample of weak sinter, medium-strength sinter, and very strong sinter.

To insure getting results that were representative of the full depth of the sintering bed, the entire sample obtained by means of the sampling basket was used as the starting material for the tumbler test. If the sample broke into pieces, all the pieces were used, regardless of their size. Results of tumbling for 10, 20, 50, 100, and 200 revolutions indicated that the median size after tumbling was proportional to the logarithm of the number of tumbler revolutions. This relation is shown in Fig. 7. After 200 revolutions, little effect on the median size of sinter resulted from one or two additional tumbles. Therefore, the 200-revolution test was selected as a standard test because it tends to minimize differences in the severity of handling the sinter between the sintering machine and the testing laboratory. Fig. 8 illustrates typical values resulting from tumbler testing entire samples procured with the sampling basket. A sample of relatively strong sinter had a median size after tumbling of 0.251 in., a sample of relatively weak sinter, 0.184 in., and a sample taken when the sintering was not progressing satisfactorily, only 0.122 in.

The results of this method of testing were easily reproducible in that adjacent samples from the same pallet gave median-size results that agreed with one another within a few thousandths of an inch.

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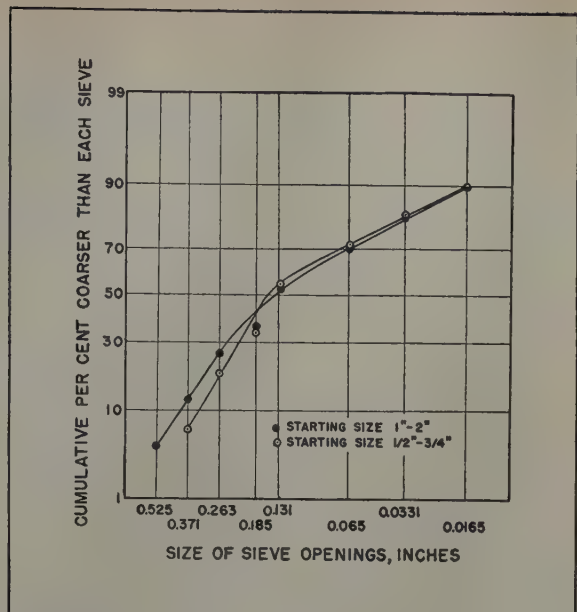
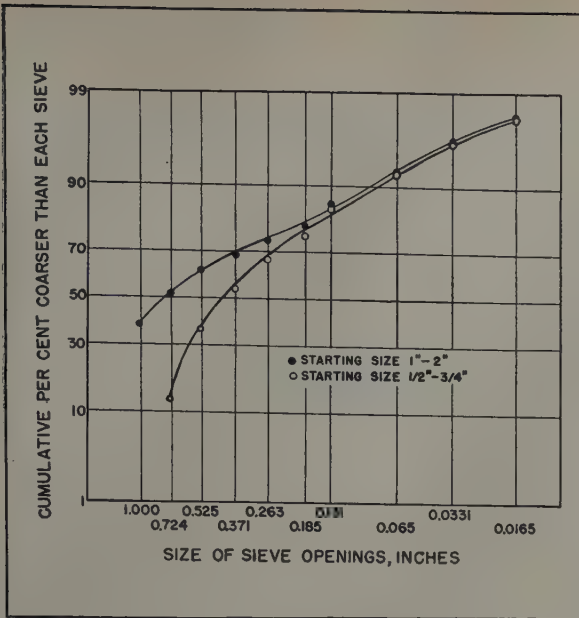


Fig. 6

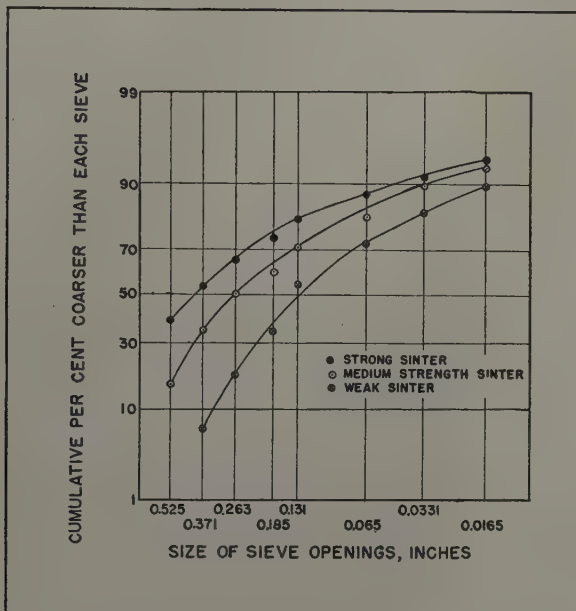
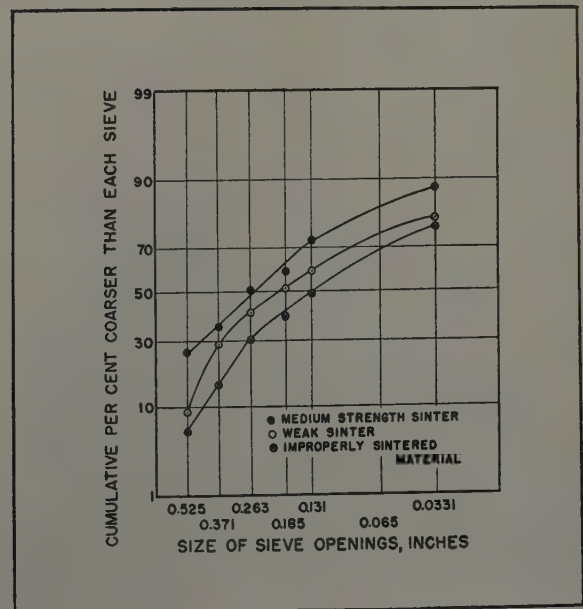
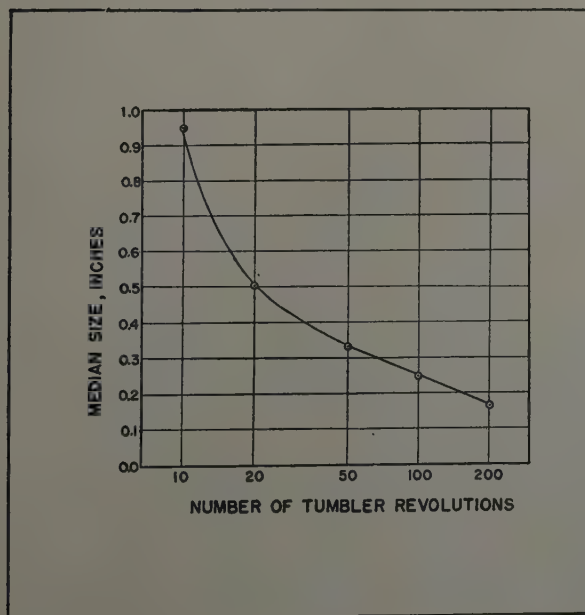


Fig. 4 indicates the effect of starting size on shatter-test results and Fig. 5 the effect on tumbler test results. Fig. 6 shows the results of sieve analysis of three different samples of sinter after tumbling for 200 revolutions, and Fig. 7 the effect of the number of tumbler revolutions on the median size of sinter samples. Fig. 8 illustrates the results of tumbler tests on samples procured with the sampling basket.

Fig. 8



The Magnetic Reflux Classifier

by Lawrence A. Roe

The magnetic reflux classifier, which utilizes the combined effects of magnetic fields and a hindered settling classifier, is a new tool for determining the quantity and quality of middlings in fine-sized magnetite concentrates. Results are given for processing a typical taconite ore, and a sketch of the apparatus is included.

IN examining magnetite ores and beneficiated products it often becomes necessary to make critical studies of the amount of grinding necessary to produce the desired degree of magnetite liberation. In the past this has been accomplished by laboratory heavy-liquid tests, which provide a method for selectively removing middling particles and free magnetite from various-sized fractions. Examination of the various products under the microscope results in fairly accurate determination of the degree of liberation. The method is quite efficient on sizes coarser than 325 mesh. Thus the heavy-liquid method of middling separation was satisfactory until the advent of present day magnetic taconite studies. When magnetite concentrates ranging from 70 to 100 pct -325 mesh are studied it becomes apparent that older methods of determining liberation size are not satisfactory and that there is need for a new method. For example, some of the low-grade magnetite ores of the Wisconsin and Michigan iron ranges require grinding to 100 pct -325 mesh to produce a magnetic concentrate containing less than 12 pct silica. Examination of concentrates from such ores often reveals that many of the middling particles consist of only very minor proportions of iron mineral.

Thus it becomes important to be able to determine the degree of grinding necessary not only for complete liberation, but also for liberation of only 80, 85, or 90 pct of the total iron mineral content. Actually, complete liberation is never attained, but is often used to designate that degree of liberation necessary for production of high-grade concentrates. A rougher concentrate, produced after elimination of a coarse-sized tailing, can usually be subjected to a second grinding stage and concentrated into a higher grade product than could be produced from the same crude ore with one stage of grinding resulting in the same overall size reduction. This fact adds to the importance of being able to determine partial degrees of liberation on any magnetite ore.

Standard laboratory methods such as heavy-liquid separation, microscopic grain counts, Davis tube magnetic separation, magnetic flocculation,

classification, flotation, and others often are not applicable, or are prohibitive because of time requirements when large numbers of fine-sized magnetite samples are investigated. The Davis tube magnetic separator is an efficient tool to use in rejecting the non-magnetic mineral particles from an ore sample. The middlings discarded by the tube separator usually are so low in iron content that they can be considered relatively unimportant in liberation studies. This condition is caused by the extremely high flux density used in the Davis tube. This flux density ranges from four to eight times the flux density produced by most of the powerful commercial machines in use today. Thus the problem resolves itself into a search for a method of selectively removing middlings from Davis tube magnetic concentrates which will be both rapid and efficient.

Those methods showing most promise in the development of a process for isolating middlings from extremely fine-sized magnetic concentrates were flotation and magnetic flocculation. The use of flotation to remove middlings from magnetic concentrates is reported in the literature.^{1,2} The flotation process is effective in removing middlings from a magnetite concentrate, but physical entrapment of fine-sized free magnetite in the silica-bearing froth is an undesirable feature. The flotation method of removing middlings requires time, effort, and precise control of many variables, and does not meet the required degree of middling isolation.

Magnetic Flocculation

Magnetic flocculation has long been resorted to³⁻⁶ in efforts to upgrade magnetite concentrates. One of the new magnetic taconite plants now under construction on the Mesabi Range includes magnetic flocculation in the flowsheet⁴ as an accessory process to remove high-silica middlings and free silica which has been mechanically entrapped in magnetite flocs. The use of magnetic flocculation as a laboratory method of making precise separation of middlings was further investigated, since it offered a rapid, simple method of accomplishing the desired result.

Magnetic flocculation involves the subjection of a magnetic concentrate to a strong magnetic field, passing the concentrate in a highly flocculated condition to a hydroseparator or other classifiers of various types, and removing free silica and middlings as overflow products. In an attempt to utilize simple

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Discussion on this paper, TP 35048, may be sent (2 copies) to AIME before May 31, 1953. Manuscript, Aug. 4, 1952. Chicago Meeting, September 1952.

magnetic flocculation as a method of removing middlings from a magnetic concentrate, it soon becomes apparent that the method as generally practiced is efficient only in separating those middling particles containing very minor quantities of magnetite. Middling particles containing over 10 to 20 pct by weight of magnetite are strongly attracted to the magnetic flocs and become entrapped in a maze of magnetite particles, all strongly attracted to one another. If the velocity of the upward moving current of water is increased beyond a critical point, the magnetic flocs are torn apart to some extent, and while more middlings are released, a considerable quantity of fine magnetite is released and lost in the overflow product.

The Magnetic-Reflux Classifier

The solution to the problem was found in imposing a controllable magnetic field near the top of a hindered-settling classifier tube. The classifier then became a magnetic-reflux classifier. Fig. 1 is a sketch of one version of the apparatus now in use.

In chemical technology a reflux condenser is defined as "a condenser which continuously returns the condensed vapor to the still." Thus a magnetic-reflux classifier can be defined as a classifier which "continuously recycles or returns magnetically flocculated particles to the upward moving water currents or teeter column."

When a sample of low-grade magnetite concentrate is placed in the classifier tube the middling particles can be removed readily by controlling two variables: intensity of the magnetic field, and velocity of the upward current of water. These variables are the same controlling factors used in most wet magnetic separators, but it must be realized that there is a great difference in the degree of water control in a laboratory hindered-settling classifier tube as compared to water control in any magnetic separator in use today. In some cases it is advantageous to provide additional stirring or dispersing action by use of an AC field which operates intermittently, or a demagnetizing coil near the lower end of the classifier tube.

It has been stated by E. W. Davis of the University of Minnesota Mines Experiment Station that "the tendency for all magnetic particles to remain in the concentrate whether or not they are attached to particles of gangue is characteristic of all direct current magnetic separators when concentrating finely crushed ore. While this tendency can be counteracted to some extent by the use of agitation, with no direct current magnetic machine now in use is it possible to fraction accurately a sample of finely crushed magnetic taconite into products of different iron assay such as can be done by gravity methods using tables, for example."⁷ Eketorp also reports on the inefficiency of present DC separation on fine-sized ores.⁸

It is well known that when a classifier is operated to emphasize the differences in specific gravity of the minerals being processed, hindered-settling conditions are preferred. In such cases the classifier not only sizes the materials involved, but concentrates the heavier minerals by a sorting process. This is not true when free-settling conditions are used. Classifier tubes used in laboratory experiments had an ID of 1.80 in. and the constriction plates had holes varying from 0.036 to 0.054 in. in diam. Use of an accurately calibrated manometer on the hydraulic water line assured precise water control to

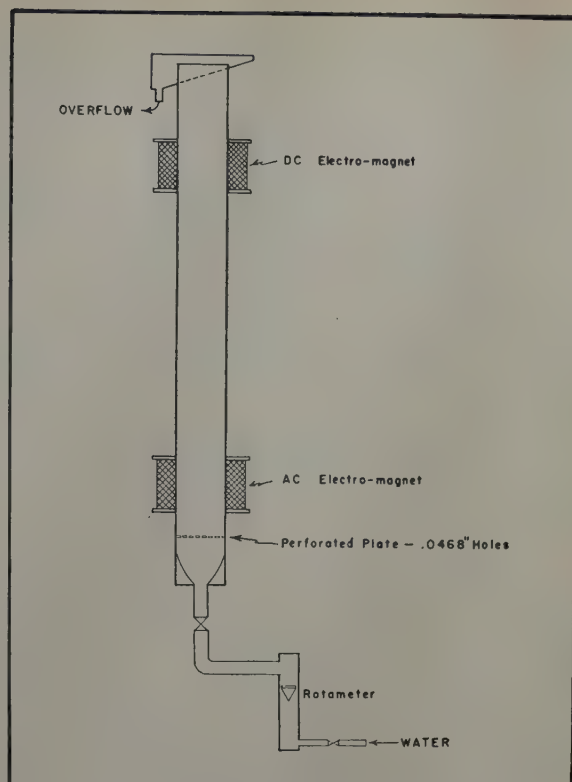


Fig. 1—Sketch of magnetic-reflux classifier for batch tests.

the classifier. High-powered photographers' spotlights were used in studying teeter column conditions in the classifier tube.

The magnetic field imposed on the classifier was provided by a 110-v DC electromagnet. The field of the electromagnet could be varied to produce flux densities from 5 gauss to 300 gauss at the internal surface of the tube wall. This field intensity is very low as compared to the range of 4200 to 14,000 gauss in the Davis tube separator.

Operating Procedures

The use of hindered-settling conditions plus precise magnetic flocculation control results in a laboratory tool capable of separating numerous fractions of magnetite middlings from magnetite concentrates. A typical operational procedure for batch tests with one type of magnetic-reflux classifier used in early tests was:

1—With hydraulic water set at 1700 cu cm per min and field magnet set at high intensity, a 50 to 100-g sample is introduced; all the sample is poured into the tube before the water level reaches the magnetic field level of the tube.

2—Period *a* begins, during which high-silica, low-iron middlings are removed as overflow product.

3—At end of period *a* the magnetic field is adjusted to medium intensity; hydraulic water remains at same flow.

4—Period *b* begins, during which middlings containing nearly equal percentages of silica and magnetite are removed as overflow product.

5—At the end of period *b*, the magnetic field is adjusted to low intensity; hydraulic water remains at same flow.

6—Period *c* begins, during which high-iron, low-silica middlings are removed as overflow product.

7—At end of period *c*, the final magnetite concentrates are withdrawn as classifier underflow.

It is evident that different ores and various degrees of grinding for the same ore may require different magnetic field strengths and different hydraulic water flows, but the proper settings are usually quickly arrived at by preliminary runs.

Test Results

Fig. 2 illustrates the appearance of the flocculated magnetite in the magnetic-reflux classifier tube during each of the three periods described under operating procedures.

The magnetite concentrate being tested was from an ore which had been ground through 100 mesh. Table I gives the chemical analysis of the classifier products.

Table I. Chemical Analysis of Classifier Products
—100-Mesh Feed

Product	Wt, Pct	Analysis, Pct		Dist., Pct	
		Fe	SiO ₂	Fe	SiO ₂
Feed	100.0	54.83	22.86	100.0	100.0
Overflow, period a	6.5	20.13	62.00	2.4	19.9
Overflow, period b	5.1	29.61	51.38	2.8	13.0
Overflow, period c	7.5	43.18	33.04	5.9	12.2
Underflow	80.9	60.29	13.74	88.9	54.9

Table II. Chemical Analysis of Classifier Products,
—200-Mesh Feed

Product	Wt, Pct	Analysis, Pct		Dist., Pct	
		Fe	SiO ₂	Fe	SiO ₂
Feed	100.0	57.98	16.23	100.0	100.0
Overflow, period a	5.9	24.86	55.78	2.5	20.3
Overflow, period b	6.8	37.24	40.12	4.4	16.8
Overflow, period c	4.9	51.02	24.40	4.3	7.4
Underflow	82.4	62.48	10.94	88.8	55.5

From these results it is evident that 88.9 pct of the iron was recovered as concentrate, while 45.1 pct of the silica was rejected in three distinct middling fractions. Photomicrographs of these middling fractions are shown above the photographs of the classifier tube in Fig. 2. The magnetite contents of the three middling fractions were 28, 41, and 60 pct respectively. The degree of flocculation of the respective overflow products gives visual indication of the quantity of magnetite present in each product. Photomicrograph (d) of Fig. 2 shows the highly flocculated classifier underflow product which contained 83 pct by weight of magnetite.

The results listed in Table I show that when a —100 mesh grind was used it was possible to produce a final magnetite concentrate containing 60.29 pct iron and 13.74 pct silica. Present day taconite studies are, in many cases, aimed at producing iron concentrates containing not over 12 pct silica. Another test was made on the ore previously tested, but using a —200 mesh grind. Table II lists these results.

Again, 88.8 pct of the iron was recovered as concentrate and 44.5 pct of the silica was rejected in three distinct middling fractions. However, the concentrate grade was raised to an acceptable figure: 62.48 pct iron and 10.94 pct silica.

For comparison, Table III lists the results of standard Davis tube tests, 1.80 amp, on —100, —200, and —325 mesh ore samples and the results of a Davis tube test at low amperage on —200 mesh ore. The ore was the same material used, after a magnetic concentration step, in the magnetic-reflux classifier tests.

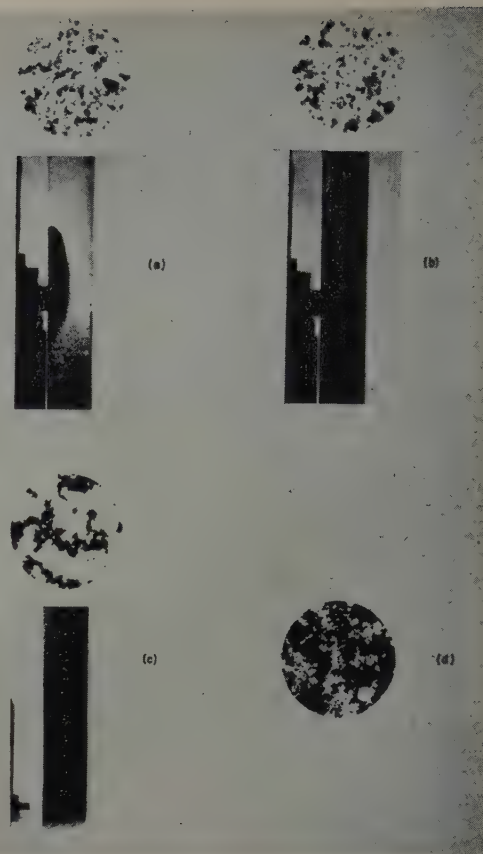


Fig. 2—Appearance of classifier tube during each of the three operating periods a, b, and c. Photomicrographs above the tube show degree of flocculation of the overflow product for each period and the underflow product, d. In this particular test the magnetic field was applied to only one side of the tube. Good results were obtained, but capacity was low.

Thus it can be seen that Davis tube tests on extremely fine-grained magnetite ores at either high or low current intensity are not satisfactory for use in liberation studies which have as their objective the rapid determination of an optimum commercial grind. Conventional tube tests show that a —325 mesh grind is necessary to produce a concentrate containing less than 12 pct silica, but no information is obtained regarding the middlings which would be produced in a commercial operation. It was previously shown, by using the magnetic-reflux classifier, that a —200 mesh grind would result in an acceptable concentrate. Also, the three middling fractions in the rougher magnetic concentrate would contain 24.86, 37.24, and 51.02 pct iron respectively.

Table III. Davis Tube Tests

Product	Mesh	Davis Tube Amperage	Analysis, Pct		Dist., Pct	
			Fe	SiO ₂	Fe as Magnetite	SiO ₂
Feed	—	—	35.82	43.22	95.0	100.0
Magnetic Concentrate	—100	1.80	54.83	22.86	99.0	30.7
Magnetic Concentrate	—200	1.80	57.98	16.23	98.0	22.2
Magnetic Concentrate	—325	1.80	62.89	11.16	98.0	14.4
Magnetic Concentrate	—200	0.20	62.78	11.06	47.9	6.6

The last two fractions, representing 11.7 pct of the weight and 8.7 pct of the iron in the rougher concentrate, would in all probability be reground and recirculated with new feed to the plant. When —200 mesh ore was processed at low amperage in the

Davis tube, an acceptable concentrate resulted, but less than 50 pct of the magnetite reported in the concentrate. When the non-magnetic fraction is reprocessed at successively higher amperages, a series of middling products can be obtained, but the sensitivity of the separation is far too low to enable the segregation of precisely graded middling fractions, and it is difficult to extrapolate how much of the low-silica, high-iron middlings can be added to the 0.20 ampere-concentrate and still not exceed the 12 pct silica figure. At this point it would be well to mention that the Davis tube is a useful tool for studying liberation characteristics of coarse-grained magnetite ores. For the purposes of this paper, coarse-grained ores are those which require grinding not finer than 65 mesh to liberate most of the magnetite. In such ores the extremely fine-sized middling particles are relatively unimportant and thus need not be isolated for study in most cases.

The magnetic-reflux classifier is the subject of a patent application. Interest in the selective separation of middlings from magnetic taconites is running high, and new types of magnetic separators involving principles exemplified by the magnetic reflux classifier may materialize in the future.

Acknowledgment

The original work on the magnetic-reflux classifier was done by the author at the Ore Research Laboratories of Jones & Laughlin Steel Corp., and

this work has been continued at the Metallurgical Laboratories of Bjorksten Research Laboratories, Inc., Madison, Wis. Thanks are due both of these corporations for their interest in the problems of magnetic taconite beneficiation.

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Work Indexes Tabulated

by Fred C. Bond

SIX years have passed since the last grindability table was published.¹ In that time the list has been increased with many new tests, and the development of the new Third Theory of Comminution² has made possible the reduction of all results to a consistent work index value.

In this tabulation all grindability and crushability test results are given in terms of the work index, with formulas for reversion to the older units if desired. The average values for 57 different classes of materials are listed in Table I. The total number of work index values listed is 1211, and the overall average work index is 14.42.

The complete tabulation, with each test listed alphabetically both under material type and under company name, can be obtained by writing to the Allis-Chalmers Manufacturing Co., Processing Machinery Department, Milwaukee 1, Wisconsin.

The work index W_i is the calculated total work input in kilowatt-hours per short ton required to re-

duce from theoretically infinite particle size to 80 pct passing 100 microns, or to approximately 67 pct passing 200 mesh.

The work inputs required per ton for the same size reduction of different materials are directly proportional to their work indexes, and according to the Third Theory the work input per ton is inversely proportional to the square root of the diameter of the product particles.

Where W represents the work input in kilowatt hours per ton, where F is the feed size, or diameter in microns of the square hole which 80 pct of the feed passes, P the product size, or microns which 80 pct of the product passes, and R_r the reduction ratio F/P , the work index W_i is found from

$$W_i = W \left(\frac{\sqrt{R_r}}{\sqrt{R_r} - 1} \right) \sqrt{\frac{P}{100}} \quad [1]$$

When the work index W_i is known, the work input W required to reduce from any feed size F to any product size P is found from

$$W = W_i \left(\frac{\sqrt{R_r} - 1}{\sqrt{R_r}} \right) \sqrt{\frac{100}{P}} \quad [2]$$

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Table I. Average Work Indexes

Material	Number Tested	Average	
		Specific Gravity	Work Index
All materials tested	1211		14.42
Andesite	6	2.84	18.25
Barite	7	4.50	4.73
Basalt	3	2.91	17.10
Bauxite	4	2.20	8.78
Cement clinker	14	3.15	13.56
Cement raw material	19	2.67	10.51
Coke	7	1.31	15.18
Copper ore	204	3.02	12.73
Diorite	4	2.82	20.90
Dolomite	5	2.74	11.27
Emery	4	3.48	56.70
Feldspar	8	2.59	10.80
Ferro-chrome	9	6.66	7.64
Ferro-manganese	5	6.32	8.30
Ferro-silicon	13	4.41	10.01
Flint	5	2.65	26.16
Fluorspar	5	3.01	8.91
Gabbro	4	2.83	18.45
Glass	4	2.58	12.31
Gneiss	3	2.71	20.13
Gold ore	197	2.81	14.93
Granite	36	2.66	15.05
Graphite	6	1.75	43.56
Gravel	15	2.66	16.06
Gypsum rock	4	2.69	6.73
Iron ore			
Hematite	56	3.55	12.93
Hematite — specular	3	3.28	13.84
Oolitic	6	3.52	11.33
Magnetite	58	3.88	9.97
Taconite	55	3.54	14.60
Lead ore	8	3.45	11.73
Lead-zinc ore	12	3.54	10.57
Limestone	72	2.65	12.54
Manganese ore	12	3.53	12.20
Magnetite	9	3.06	11.13
Molybdenum ore	6	2.70	12.80
Nickel ore	8	3.28	13.65
Oil shale	9	1.84	15.84
Phosphate rock	17	2.74	9.92
Potash ore	8	2.40	8.05
Pyrite, ore	6	4.06	8.93
Pyrrhotite, ore	3	4.04	9.57
Quartzite	8	2.68	9.58
Quartz	13	2.65	13.57
Rutile ore	4	2.80	12.68
Shale	9	2.63	15.87
Silica sand	5	2.67	14.10
Silicon carbide	3	2.75	25.87
Slag	12	2.83	9.39
Slate	2	2.57	14.30
Sodium silicate	3	2.10	13.50
Spodumene ore	3	2.79	10.37
Syenite	3	2.73	13.13
Tin ore	8	3.95	10.90
Titanium ore	14	4.01	12.33
Trap rock	17	2.87	19.32
Zinc ore	12	3.64	11.56

Eq. 2 is used to find the work input required for any reduction size in dry crushing or wet grinding directly from the listed work index; the work input required for dry grinding is found by multiplying the work for wet grinding by 4/3.

The 80 pct passing size in microns is a convenient term for expressing the fineness of a crushed or ground product, and a convenient base for calculation of the reduction ratio and the work required for reduction.³ It is readily found by plotting the percent passing size distribution curve.

When the size distribution curves of the feed and product are parallel the reduction ratio remains constant for all particle sizes, and the work index calculated from the 80 pct passing size is equivalent to that calculated on the basis of any other selected percent passing size. Small differences between the slopes of the plotted feed and product lines have only a slight effect on the work index, proportional to the square root of the effect upon the surface areas. However, when the feed has had the fines removed the equivalent feed size is increased and the work index may be considerably in error. In cases where a crusher feed is scalped by passing over a grizzly screen the tonnage and size distribution of the fines removed are rarely known, and it is preferable to consider the feed to the grizzly as

being equivalent to the feed to the crusher and to calculate the work index on this basis.

The listed work index values calculated from laboratory grindability tests apply directly to a wet grinding overflow type of rod mill 6 ft in diam in open circuit, and to a wet grinding overflow type of ball mill 7½ ft in diam in closed circuit with a rake classifier at 250 pct circulating load, and with 80 pct or more of the feed passing 4 mesh. Correction factors should be applied for oversize feed and other operating conditions which change the grinding efficiency. When work index values at several different product sizes are available the value nearest the actual product size should be used. The work index represents the total work input to the reduction machine necessary in installations of average efficiency.

Where C represents the impact crushing strength¹ in foot-pounds per inch of thickness, and S is the specific gravity, the work index is

$$Wi = 2.59 C/S \quad [3]$$

and the impact crushing strength is found from the listed work index by

$$C = Wi S/2.59 \quad [4]$$

Where Grp represents the rod mill grindability¹ in net grams produced per revolution of the test mill which pass a sieve opening of P_1 microns, the work index is found from

$$Wi = \left(\frac{16.6}{Grp^{0.625}} \right) \sqrt{\frac{P_1}{100}} \quad [5]$$

and the rod mill grindability is found from the listed work index by

$$Grp = \left[\left(\frac{16.6}{Wi} \right) \sqrt{\frac{P_1}{100}} \right]^{1.60} \quad [6]$$

Where Gbp represents the ball mill grindability at P_1 microns the work index is found from

$$Wi = \left(\frac{16}{Gbp^{0.820}} \right) \sqrt{\frac{P_1}{100}} \quad [7]$$

and the ball mill grindability is found from the listed work index by

$$Gbp = \left[\left(\frac{16}{Wi} \right) \sqrt{\frac{P_1}{100}} \right]^{1.22} \quad [8]$$

The work index can be calculated by Eq. 1 from commercial crushing or grinding data or from pilot mill tests, and compared with the laboratory work index to obtain the relative mechanical efficiency.

The capacity of any size-reduction machine over its normal range of feed and product sizes is proportional to $\sqrt{Rr} \times P / (\sqrt{Rr} - 1)$. In cases where the capacity is found to vary more than this amount some condition causing inefficient operation should be suspected. These may include packing in a crusher, oversize feed or improper ball and rod sizes in a tumbling mill, and coating in dry grinding.

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Geophysical Case History, Fredericktown Lead District, Missouri

by Harold Powers, LeRoy Scharon, and Carl Tolman

THIS paper presents geophysical and subsurface data observed in the vicinity of Shafts No. 1 and 5 of the National Lead Co. lead mines at Fredericktown, Madison County, Missouri, see Fig. 1. The area selected is an excellent example to illustrate the close correlation between geophysical data and geological facts as revealed by drilling and mining. Such close correlations make geophysical applications appropriate to the development of known orebodies as well as the exploration for new orebodies.

In the past, magnetic and shallow electrical resistivity observations were made over limited areas in the Fredericktown lead district by the Missouri Geological Survey, with favorable results. The first extensive magnetic and electrical resistivity surveys were started by the U. S. Bureau of Mines in May 1945 in two areas located northwest and southwest of Fredericktown, as shown in Fig. 1. The results of these surveys were reported in April 1948 at Denver, Colo. Late in 1945 spontaneous-polarization surveys were made over the property of the St. Louis Smelting and Refining Co. at Fredericktown by Sherwin F. Kelly Geophysical Services, Inc. Aeromagnetic surveys were completed in 1948 over the Fredericktown quadrangle by the U. S. Geological Survey.

Since 1945 geophysical surveys have been an important part of exploration programs in the Fred-

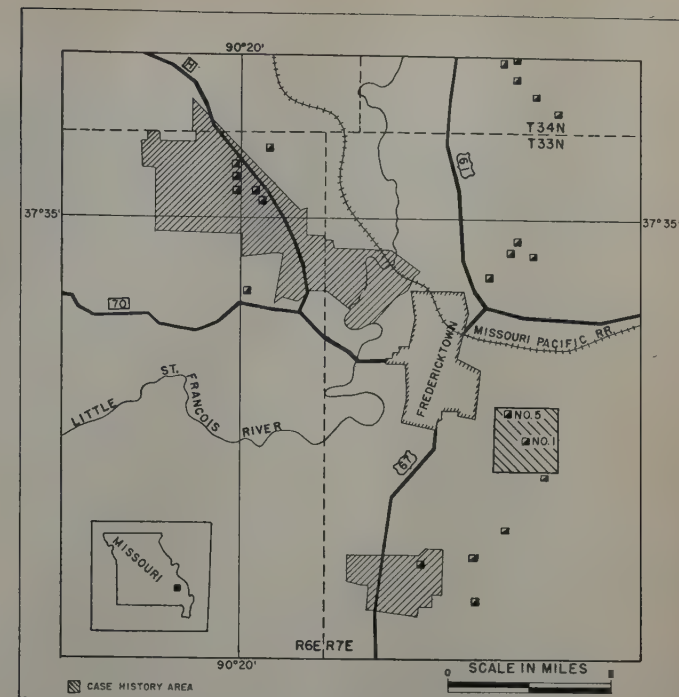


Fig. 1—Index map showing location of case history area, Fredericktown lead district, Missouri.

ericktown district. Such methods, magnetic and electrical resistivity, are now serving as guides in the further development of known orebodies.

On an irregular pre-Cambrian igneous erosion surface Lamotte sandstone of upper Cambrian age was deposited, partially filling the topographic basins, see Fig. 2. Above and conformable with the Lamotte sandstone lies the Bonnetterre formation, a dolomite with locally interbedded thin dolomitic limestones and shales at its base. This formation also only partially filled the topographic basins. The Bonnetterre was overlain by the Davis, Derby, Doe Run, and Potosi formations, which are principally dolomite.

Uplift, tilt, and slight warping has allowed erosion and weathering to remove most of the sediments overlying the Bonnetterre, and in many places, not far from the mine, the Lamotte sandstone is exposed. This erosion has also exposed areas of pre-Cambrian igneous rock.

The Lamotte sandstone, having been deposited on an irregular pre-Cambrian surface, is extremely variable in thickness and ranges from zero where it laps the pre-Cambrian rocks to 250 ft thick in the deeper basins. In the vicinity of the mines the Bonnetterre dolomite ranges from zero where it wedges out against the pre-Cambrian igneous rocks to 400 ft thick.

The ore deposits occur as a bedded disseminated type¹ and are chiefly composed of galena and pyrite with minor amounts of chalcopyrite and siegenite. Heavy pyrite and marcasite mineralization may be found throughout the Bonnetterre formation. However, there is little or no apparent association of these iron sulphide bodies lying above the ore hori-

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Discussion on this paper, TP 3507L, may be sent (2 copies) to AIME before May 31, 1953. Manuscript, Feb. 20, 1952. New York Meeting, February 1952.

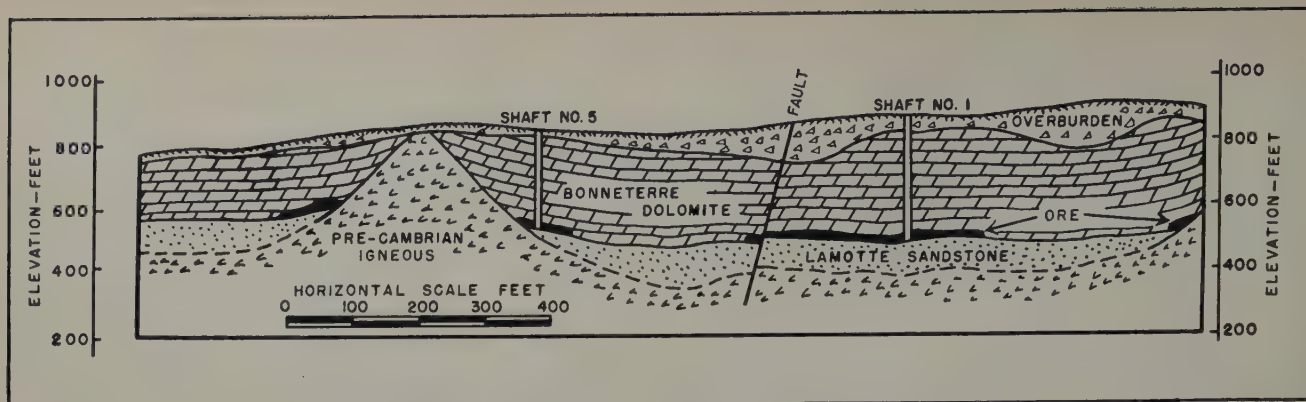


Fig. 2—Generalized geologic section, Fredericktown lead district, Missouri.

zon and the ore deposits. Economic mineralization is characteristically confined to the lower 20 to 25 ft of the Bonneterre dolomite. The average depth from surface to this favorable zone is 300 to 400 ft.

For prospecting and development purposes, the orebodies can be divided into two types. One type characteristically occurs near the Lamotte sandstone pinch-out against the pre-Cambrian igneous. The other type is in areas of unusual Lamotte sandstone highs and lows. The erratic contour of the Lamotte sandstone may have been caused by sand being deposited on an irregular pre-Cambrian surface, by the formation of sandbars at the time of deposition, or by tectonic disturbances.

The host rock near the Lamotte sand pinch-out line has been favorably prepared by a series of fractures including joints and minor faults which parallel the Lamotte pinch-out line and often coincide with it. The host rock in areas of Lamotte sandstone highs and lows is structurally prepared in much the same manner. Within the area at least one fault is believed to displace the pre-Cambrian igneous rock.

There are two known localized sites for ore deposition within the immediate mine area, one being the lower 20 to 25 ft of the Bonneterre dolomite near the Lamotte sandstone pinch-out line and the other

being the lower 20 to 25 ft of the Bonneterre dolomite in areas of Lamotte sandstone highs and lows. Both localized areas appear in some measure to be associated with the pre-Cambrian topography.

An area of approximately 370 acres in the vicinity of Shafts 1 and 5 was chosen to ascertain the practicability of using geophysical methods as aids in the development of ore reserves. Magnetic, spontaneous polarization and resistivity surveys were conducted as described in detail below.

Although spontaneous polarization measurements did not prove practical in assisting in the location of new orebodies or extending known ones in the Fredericktown district, magnetic and resistivity surveys have led to a more intelligent development of ore reserves by actual drilling. Magnetic surveys revealed the presence of buried igneous knobs and ridges or the subsurface extensions of those already known. After the location of such igneous masses had been established, electrical resistivity measurements helped in determining the depth to the igneous surface, the depths to the contacts of the various sedimentary horizons involved, and the location of zones of fracture.

Magnetic Surveys: Measurements of vertical magnetic intensity in the area were done on a 100-ft

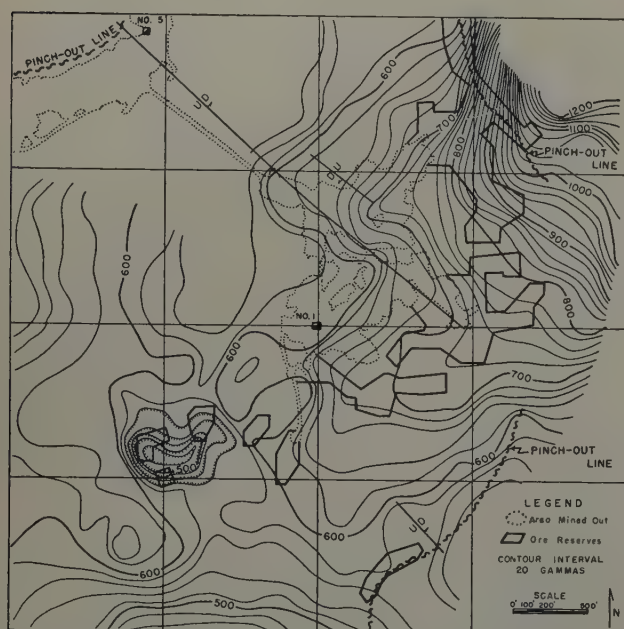


Fig. 3—Vertical intensity magnetic map showing mined-out areas and ore areas.

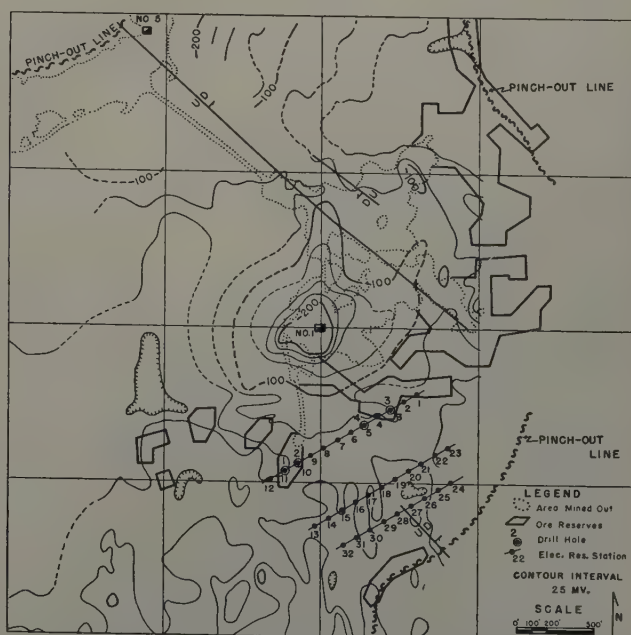


Fig. 4—Self-potential map showing locations of electrical resistivity depth stations.

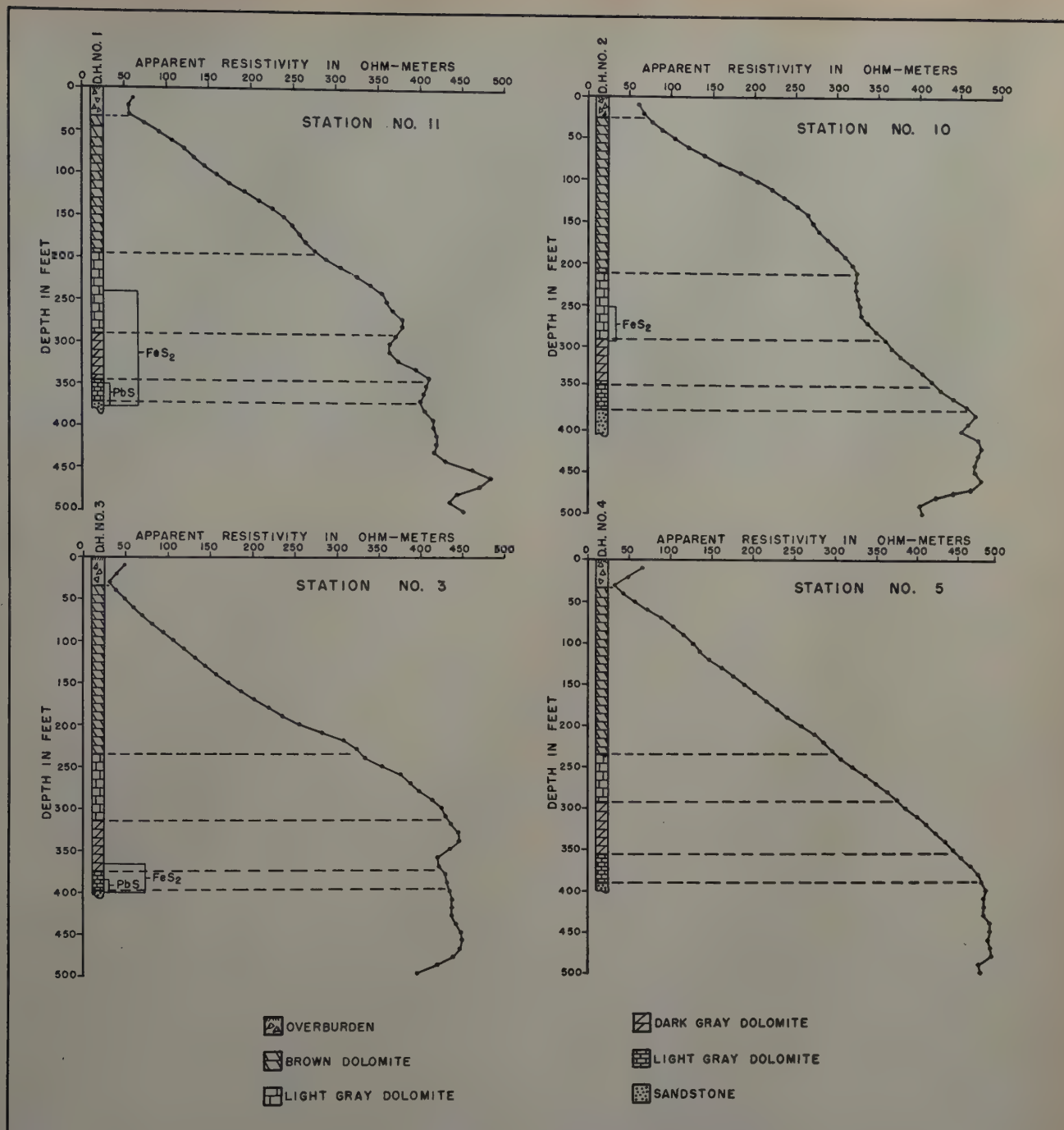


Fig. 5—Apparent resistivity profiles correlated with drill log data.

grid using an Askania-type vertical intensity magnetometer with a scale constant of 20 gammas. From these observed data an isogam map, Fig. 3, having a contour interval of 20 gammas has been prepared and superimposed on a structural map showing mined ore, proved ore, and the pinch-out line. Measurements could not be made in the vicinity of Shaft No. 5 because of surface interference.

Considerable variation in magnitude and trend of the magnetic lines appears to be correlated with known geological facts. For example, a magnetic maximum of 1200 gammas is associated with the buried igneous mass northeast of Shaft No. 1, decreasing in a southwest direction toward the shaft. This suggests that the surface of the igneous mass dips southwest in the form of a ridge and conforms with a sand high in the same area. Then too, the

isogam lines appear to be associated with the major fault located north of Shaft No. 1. This association is best observed by following the 600-gamma contour line. It trends southwest to the fault, which it parallels southeast for 900 ft and then southwest past Shaft No. 1. Ore was of good grade in the vicinity of the fault. Furthermore, the magnetics disclosed a likelihood of structural features in the igneous basement southwest of Shaft No. 1 centering around the magnetic low of 480 gammas. In this area it is observed that the 600-gamma contour lines block this low on three sides; the existence of faulting similar to that mapped in the mined-out area to the northeast is therefore assumed. On the basis of this interpretation and the fact that ore was mined in relation to the major fault it was recommended that drilling be concentrated around the flanks of

this magnetic low. Such drilling has revealed three areas of ore grade. A sand low which also indicates faulting exists in approximately the same area. Finally, the magnetics reveal a high in the basement rock in the extreme southwest corner of the area. Subsequent drilling revealed a sand high, but only low grade mineralization. To summarize, it can be said that in detail magnetics are an indirect guide for delineating areas in which to concentrate drilling.

Spontaneous Polarization Surveys: Late in 1945 Kelly² made a spontaneous polarization survey which included this area. This method depends upon measuring at the ground surface weak electrical currents generated by sulphide deposits. Over such deposits potentials of a few tens to several hundred millivolts may be observed. In this area readings were made along east-west parallel traverses spaced at 200-ft intervals. Readings were usually taken at 50-ft intervals along these lines, although in some places this interval was reduced to 25 ft. From these observed data a self-potential map with a contour interval of 25 mv was prepared by Kelly and is reproduced in Fig. 4.

A striking anomaly of 100 mv is observed near the No. 1 Shaft and in the general vicinity of the No. 1 orebody. However, it is believed that this anomaly is due to surface conditions. There was no marked indication of other areas over which drilling could be recommended. It may be observed that any anomaly measured by this method is produced by the electro-chemical activity of pyrite located in the oxidizing zone and not by the sulphides in the ore horizon, which lies below the water table.

Electrical Resistivity Surveys: At the time of the electrical surveys the orebodies now established south of Shaft No. 1 were not known. The southern pinch-out line had been fully explored by drilling which revealed the orebody on the southern end. In the area between the southern pinch-out line and the mined-out area south of Shaft No. 1 electrical resistivity measurements were made to guide drilling rather than to drill on a grid system.

The electrical resistivity method, using the Gish-Rooney type of equipment³ and the Wenner⁴ electrode configuration, was employed. Three traverses were established, see Fig. 4, with 32 resistivity station centers at 100-ft intervals. Each station was observed to a depth of 500 ft, a measuring interval, a , of 10 ft being used.

The electrical data was plotted as a function of the measuring interval. Two sites were recommended as most likely to bear conditions suitable for mineralization. However, other drilling was also carried on to test the interpretations fully.

Four apparent resistivity profiles representative of the types measured, with resultant drilling data, are shown correlated in Fig. 5. Drilling was recommended and carried out at these locations and the data compared with the observed resistivities. These

calibrations, which are significant, were used to interpret the entire electrical survey. Very definite correlations are shown at the overburden-dolomite and dolomite-sandstone contacts. Intermediate zones of the Bonnetterre are not readily correlated. Only two holes, 1 and 3, logged good ore grade mineralization. These ore zones are reflected by the decrease of resistivity, Stations 11 and 3, just above the sandstone in the lower light gray member of the Bonnetterre. At Station 10 a decrease in resistivity occurs at a depth of 210 ft. This decrease is too shallow to represent galena mineralization but probably reflects the heavy iron sulphides and fractured ground encountered in drilling. A similar decrease was measured at Station 11 at a depth of 290 ft, again corresponding to heavy iron sulphide mineralization and fractured rock. Where the dolomite is fairly uniform, Station 5, the apparent resistivity profile shows a continuous rate of increase until it reaches the sandstone contact. Drilling at Station 5 and in areas of similar resistivity curves revealed neither iron nor lead sulphide mineralization. Hence the electrical resistivity results do indicate areas which are structurally favorable for mineralization and can be used to guide further drilling. But the method does not directly reveal galena mineralization.

This electrical resistivity work not only established the limits of the orebody south of Shaft No. 1 but also resulted in a considerable saving of drilling costs by eliminating the need for pattern drilling.

Conclusions

The use of magnetic and electrical resistivity prospecting in the Fredericktown lead district has long been established as part of the program for extending and exploring new ore zones. When taken in close coordination with geologic data available, general geologic conditions can be recognized and used intelligently to develop orebodies further. Considerable savings in costs are effected through drilling guidance and reduction of the drilling footage normally required. It is understood that neither magnetic nor electrical resistivity methods give a direct indication of ore, but when properly used, in the light of good geologic mapping, they do indicate conditions favorable for mineralization.

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- ³ O. H. Gish: Improved Equipment for Measuring Earth-Current Potentials and Earth-Resistivity. *Bull., National Research Council*, Vol. 2, Part 2 (November 1926) p. 86.
- ⁴ Frank Wenner: A Method of Measuring Earth Resistivity. *Bull., National Bureau Standards*, Vol. 12, No. 3, 1916, pp. 469-482.

Correction

In the October 1952 issue: TP 3385F. Application of Coarse Coal Magnetite Separators in an Existing Circuit. By V. D. Hanson, W. K. Heinlein, and J. M. Vonfeld. P. 985, col. 2, par. 4, should read: Magnetite consumption for 1951 at Champion was 0.8 lb per ton of feed, or, since the combined recovery is 53 pct, the consumption per clean coal ton was 1.5 lb.



Haulage System in St. Joseph Lead Co. Mines of Southeast Missouri

by E. A. Jones

THE Southeast Missouri division of the St. Joseph Lead Co. normally hauls and hoists over 5 million tons of lead ore each year. This ore is mined in the stopes and headings of 20 mines, hauled to a main line system, see above, and transported up to 6.4 miles to ore-hoisting shafts. Average haulage distance is 2.16 miles. A few years ago a distinguished mining engineer visiting this area for the first time remarked, "You do not have a mining problem; yours is a haulage problem." Its relative importance may be gaged by the fact that mine haulage represents about 25 pct of the total cost of mining ore, the most costly single underground operation.

The Southeast Missouri lead belt is a mining district located about 65 miles south of St. Louis. This district is roughly 6 miles wide and 10 miles long. The ore horizon lies about 400 ft below the surface where most of the ore is mined from flat-lying beds of dolomitic limestone.

This paper deals with the central part of the district, namely, that portion which contains a group of interconnected mines all in a central haulage system, serving four hoisting shafts. The two major

hoisting operations are carried out at Federal No. 17 shaft and Leadwood No. 12 shaft. No. 17 accounted for about 70 pct of the ore hoisted in the district during 1950, averaging about 14,000 tons per operating day. The underground dumping and skip-pocket system at this shaft is similar to those of the other shafts except that it has two rotary dumps. The other ore-hoisting shafts have only one. It might be mentioned that waste rock is hoisted at still other shafts, hauled off, and dumped on the surface by trucks.

The hoisting shafts are on a connected system of about 200 miles of underground 24-in. gage main line railroad. There is also the Bonne Terre system, with its own ore-hoisting shaft, which consists of 13.5 miles of 24-in. gage main line, located in the extreme north portion of the lead belt area. Smaller outlying mines such as Doe Run No. 12 and Hayden Creek operate on trackless or shuttle car haulage.

In addition to the system of main line haulage, each contributing mine group has its own extensive system of stope or heading railroad. Trackless mining is making considerable headway in the Leadwood and Desloge mines. Here the shovel loads into a shuttle car which in turn dumps into mine cars spotted on the main line track system.

When the merits of underground haulage and surface transportation are compared, it must be remembered that in early days there were many individual

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Discussion on this paper, TP 3437A, may be sent (2 copies) to AIME before June 30, 1953. Manuscript, Dec. 28, 1951. New York Meeting, February 1952.

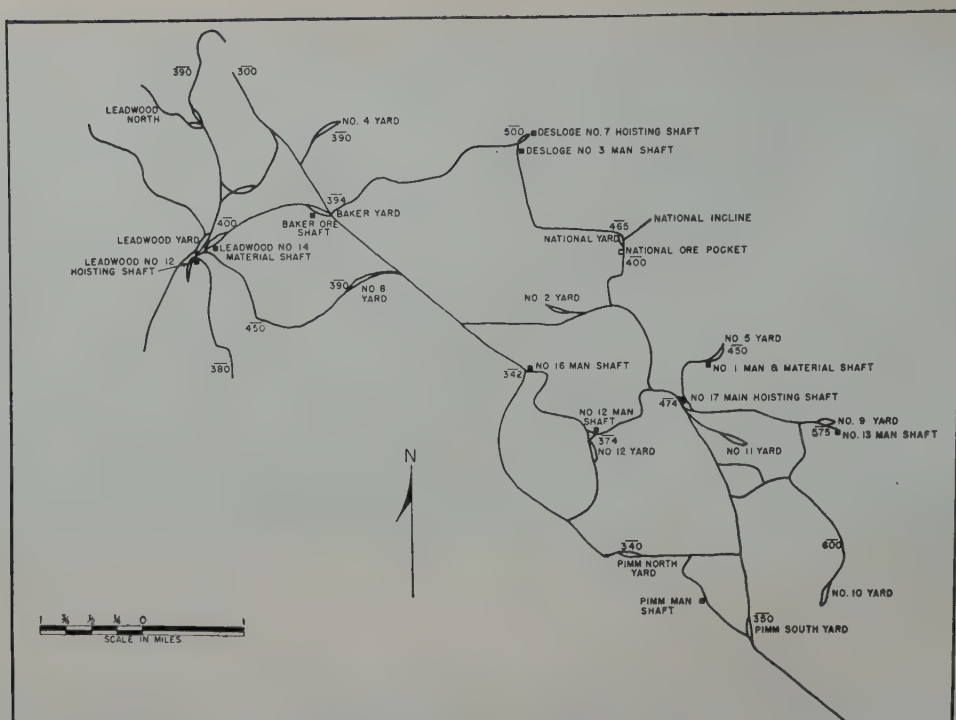


Fig. 1—A diagram of the main line haulage system showing the location of various shafts, the National Incline, and some of the main gathering yards.

mines, each with its ore-hoisting shaft from which the ore was hauled by surface railroad to a mill. As the mines all came under one company operation, it was found more economical to haul to a central shaft where the ore could be hoisted and dumped directly into the mill bins. The mining faces advanced farther and farther from the shafts; consequently there were longer and longer hauls. Tonnage increased with length of haul, and there was a smaller proportionate increase in cost per ton-mile.

An economic limit is reached when the combined costs of drift, track, and trolley exceed the cost of sinking a new shaft, with the attendant expenses of road building and truck transportation. The establishment of this limit is a problem to be worked out in detail for each orebody.

As the production of ore has increased problems have developed and bottlenecks have occurred. These have been met by improving the main lines with reduced grades, increasing radius of curves, and increasing the size of locomotives and cars. Other troubles have been solved by the use of electric switch-throwing devices and traffic signal systems, and by the installation of automatic signals and radio-telephones in congested areas.

The Haulage System

Flexibility in the main line underground haulage system allows for transportation of ore, supplies, or men to Federal No. 17, Leadwood No. 12, Baker, or Desloge No. 7 from any point on the track of the contributing mines. Many other man and supply shafts are easily accessible, see Fig. 1. In the near future, the National Incline will be ready to handle supplies for all the mines connected to the main line system, which serves 20 mine groups and the two general mines, each under a mine captain. Each mine group has its own haulage system and rolling stock.

The General Mines: The two general mines, Federal No. 17 and Leadwood No. 12, are equipped with extensive shop facilities as well as switching, dumping, and hoisting installations. No. 17 underground

shop is one of the largest in the country and is fully equipped to overhaul and rebuild locomotives, shovels, and any other type of mining machinery used locally. Leadwood No. 12 shop, though not as large as No. 17, is well equipped and in the past overhauled and serviced locomotives, shovels, and other mining machinery. At the present time, No. 17 shop specializes in overhauling the St. Joe shovels and all locomotives. Leadwood No. 12 shop specializes in overhauling and repairing the Joy loaders and shuttle cars. The shops proper are supervised by foremen operating under the mechanical department. Track, rotary dumps, switches, skip pockets, and scales are under the supervision of a mine captain. He is responsible for the regulation of all traffic in his general mine area. Ten ore-producing mine groups and one development unit haul to No. 17. Five mine groups haul to Leadwood No. 12, while the Baker zinc ore and Federal No. 6 zinc ore is hauled to the Baker shaft, where it is hoisted to the surface and hauled by truck to the Desloge mill. Zinc ore is also hoisted at Desloge No. 7 shaft, but at present writing this shaft is still in development. Both these latter ore-hoisting shafts have only small mine shop facilities. In addition to the machine shops, large modern underground supply houses are located at both No. 17 and Leadwood No. 12.

Stope Haulage: Actual haulage operations may be divided into three main classes: heading, gathering, and main line. Heading, or stope haulage, is handled by locomotives weighing from 4 to 8 tons which push empty cars into a stope, spot them for shovel loading, and pull the loaded cars to the stope siding. They are equipped with two traction motors and a reel motor. Operation is either from trolley pole or reel. Drum controllers are used with a variety of series-parallel connections. All locomotives have hand brakes, and the newer types hydraulic brakes as well. About 110 locomotives of this class are now in operation.

The motorman, who is trained to operate the locomotives properly, spots empties for the shovel, pulls

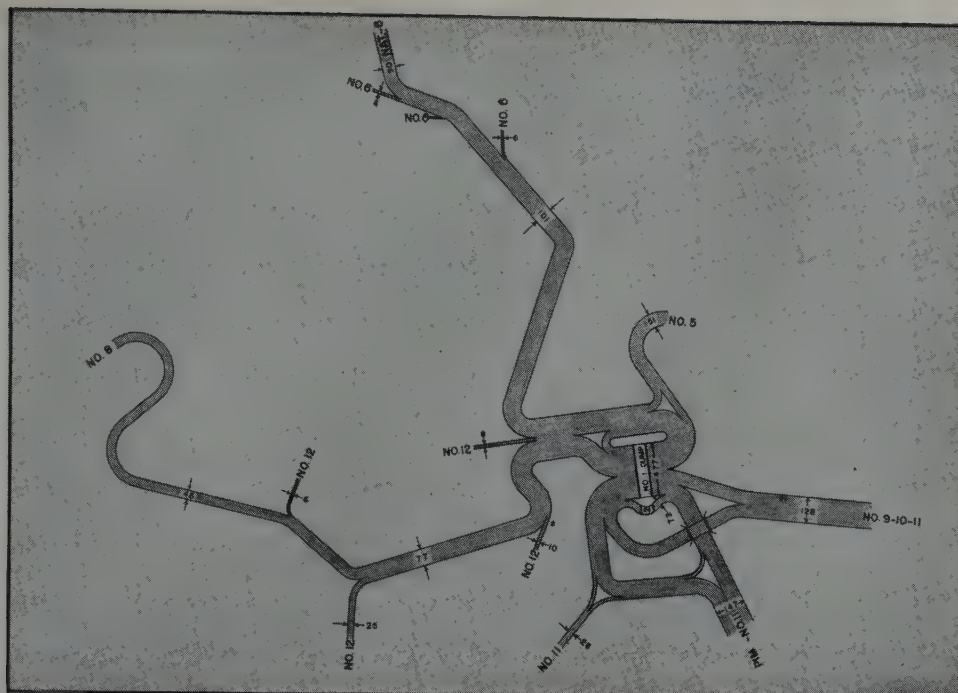


Fig. 2—A flow diagram indicating the traffic density in trains of ore dumped at No. 17 Shaft for week ending Nov. 18, 1951.

the load out to the siding, switches, and pushes in empties again. He also assists his shovel operator, making minor repairs and operating the shovel in emergency. Both members of the shovel crew share incentive bonus earnings equally; good operation is maintained because it is emphasized to the crew that additional earnings can be made from machines with few breakdowns. The captain tells the motorman the maximum loaded cars to move in a train.

Gathering and Main Line Haulage: Gathering and main line haulage may be discussed together since both use the same type of locomotive, ranging from 13 to 25 tons in weight. Gathering consists of moving the loaded cars from the stope siding to the mine assembly yard. Main-line haulage involves movement from the assembly yard to one of the central hoisting shafts. A mine group located in the immediate vicinity of a hoisting shaft may combine these two into one movement.

Nearly all locomotives have been rebuilt and re-designed and may be 2 or 3 tons heavier than when originally received from the manufacturer. They have two traction motors, and all but one use direct connected drum controllers. All have the dynamic braking system, and with the exception of a 25-ton G. E., all have screw-type hand brakes. The 25-ton G. E. has air brakes. Standard accessories on the locomotives include dimming lights, sirens, pole transfer switches, and blowers. A number are now equipped with radio-telephone sets.

As the main line tracks approach the hoisting shaft, the volume of traffic becomes proportionally greater. Since No. 17 hoists about 70 pct of the ore mined in the district, it might be well to examine its traffic flow diagram, see Fig. 2. Ore trains from the various contributing mine groups are pulled into the No. 17 General mine where they come under the control of the rotary dump operator. He regulates the actual traffic to his dump by signal lights.

Dumping and Hoisting the Ore: When an ore train is given a green light, the locomotive engineer pulls it past the dump and over the platform scales where

the loaded car weights are recorded, see Fig. 3. Then a switch is thrown and the train is backed to the rotary dump. Here, the actual car spotting is controlled by the dump operator. Three small cars are dumped at a time, or one large car, see Fig. 4. Then the locomotive engineer pulls his empty train over the platform scales where the empty car weights are recorded. The train of empties continues on a circular route until its proper main line is reached.

Unlike the other ore-hoisting shafts, No. 17 has two rotary dumps, discharging into a common skip pocket of about 2,000-tons capacity. Ore flows by gravity into the 8.5-ton capacity skip and is then hoisted about 550 ft to the surface and dumped into the Federal Mill bin.

Haulage operations around No. 17 are protected by automatic block signals for a short distance on all tracks. Beyond the automatic signals are blocks regulated by manually operated red and green signal lights all the way to the gathering loops of each mine area.

The Haulage Problem: The haulage problem varies with the demand for ore, with the location of new orebodies, with the type of ore, lead, zinc, or both, with the introduction of new types of equipment, and even with new concepts of efficiency. For example, when the price of zinc increased to the point where St. Joseph Lead Co. could mine its zinc-orebodies at a profit, it was decided to redesign the Desloge mill to concentrate zinc. It was necessary to install a skip pocket, dump, and other facilities at Desloge No. 7 shaft, and until this can be completed, the zinc ore is routed to Baker shaft.

The immediate future calls for many changes in both track and equipment. Long-range planning provides for a number of large projects not yet under way. In general, as the grade of ore decreases more tonnage must be mined, loaded, hauled, and hoisted to produce enough metal to meet demands.

The Supply Problem: In general, the supply traffic is the reverse of the ore and waste rock traffic. Supplies are lowered from the surface in supply shafts,

loaded on cars, and hauled to underground storehouses and shops. At present, all the supplies for the Federal area come down No. 1 shaft and are hauled by the supply crews to the No. 17 supply house. Each mine group makes out an order every two weeks for its anticipated needs. Then the supplies ordered are loaded into mine cars, and are picked up by a main line locomotive engineer to haul back with his train of empties.

As an illustration of the volume of supplies required for track building alone, these mines use about 100,000 white oak 6x8-in. ties, and over 2000 tons of rails of various weights and lengths per year. These figures do not include the large numbers of rails and ties taken up and re-used elsewhere throughout the mines. In many instances broken rails are cut off and a new section welded in place.

Trackless mining is making rapid advances in the Leadwood area. This results in a great reduction of the number of 30-lb rails required, for they are used only in heading track. Track building is one of the major items of underground haulage and elimination of a large portion of heading track would effect a material saving in man-hours and supplies. However, the comparative overall costs of track and trackless mining are still to be determined.

To facilitate further the handling of supplies, work is in progress to put the new National Incline in operation. It is planned to ship supplies directly from the large surface National Warehouses to any mine area without rehandling. This incline will also serve as a faster entry point for the National crews, saving about an hour per day travel time.

The Construction and Maintenance of Underground Railroads

The construction and maintenance of about 200 miles of underground railroad, with the attendant trolley and feeder systems, make up a large part of the expense charged to underground haulage. Most of the newly constructed main lines are laid with 70-lb rail, whereas the older ones were laid with 60-lb rails. Most of the frogs used are No. 5. However, No. 7 frogs are being used in some places. A large portion of this track is laid on creosoted ties. Because large volumes of water may be encountered in development operations drainage ditches are carried alongside the main line track, especially in

drifts. Keeping the track above water level makes a much better road bed, and a longer lasting track in general. Because of the age of the district, large areas have been mined out. In many cases the main line track is routed through these old mined-out areas, and laid on fills or in ditches depending on the relative elevations of track and old stope bottom. Here the creosoted tie is used to prevent loss from dry rot.

The typical main line drift is about 11 ft wide and 9½ ft high. A water ditch between 1 and 2 ft wide is generally located on one side of this drift. Signal wires and power lines are hung on one wall, and the airlines are either hung on the wall or placed on the ground on the opposite side of the drift from the water ditch. Trolley wire is installed 7 ft above the rail, and offset about 8 in. to one side of the track. It is generally accompanied by a feeder line ranging in size from 300,000 to 1,000,000 cm. With the advent of the mine radios, an antenna wire is generally installed above the trolley wire.

The Three Classes of Track: The St. Joseph track system may be divided into three classes. The first class system includes the most heavily traveled section of the main haulage line. The second class system also is a part of the main haulage system but carries far less traffic. The only differences between this and the first class system is that there are fewer treated ties and more 60-lb rails. The third class of underground railway is heading track from gathering loops to the stope headings. This track is constructed of 30-lb rails and No. 3½ or No. 4 frogs laid on untreated ties. To facilitate loading piles of broken ore much of this track is shifted either by trackmen or shovel crews. Trolley wire is extended only far enough for the reel locomotives to operate up to the shovel. Little attention is given to this track bed, and generally no bonding is used.

The main line, both first and second class, is constructed and maintained by track gangs assigned from each mine group. A gang generally consists of a track foreman and 6 to 8 trackmen. Each gang has up to 20 miles of track to maintain, working on day shift only, and all maintenance work is done under traffic unless several hundred feet of track must be relayed. Such a job is carried out either on split-shift or on Saturdays. The track gang is subject to call at any time to clean up wrecks and derailments, but most of the crew's work is track maintenance. The gang is responsible for construction of new track in drifts and other development projects, shifting and extending heading track to facilitate shovel loading, and supplying men for any odd job which may turn up.

Track Maintenance: Among the principal items of maintenance costs are drainage and keeping the track clean. If drainage ditches are not kept open, water rises under the track, the joints start pumping mud, ties begin to rot, and in a very short time the track requires a complete overhaul. Mechanization of this type of work underground is not so feasible as on the surface. Hence the pick and shovel in the hands of a man is still about the sole means of track maintenance. Mechanical track cleaners have been tried with only fair results.

The trolley and feeder systems are installed by the mine electricians, who also install the electrical signal systems, the electrical components of the automatic switches, the antenna for the radio-telephones, and any other type of wiring. However, the trackman installs the rail bonds and welds in new rail



Fig. 3—Weighing ore cars on the platform scales at No. 17 Shaft. Note speed at which the ore train is weighed. After being dumped, the train of empties will be reweighed to determine the net weight of ore per car.

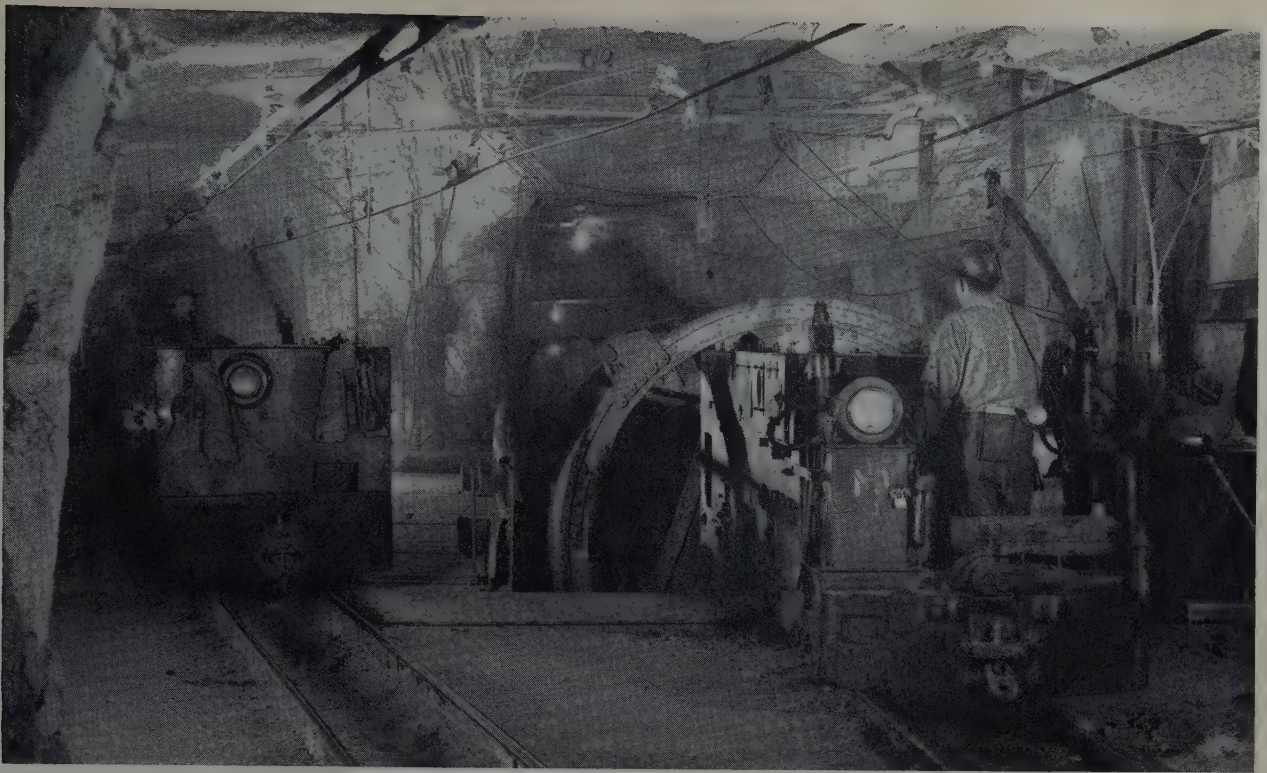


Fig. 4—Dumping a 12.5-ton car in one of the No. 17 rotary dumps. Note the two 15-ton locomotives and the wiring systems. Actual car spotting is controlled by the dump operator. Three small cars can be dumped, or one large car, as shown here.

pieces and angle bars. There are differences of opinion concerning the relative efficiencies of the copper wedge type of bond and the welded joint.

It is difficult to assess the value of a high standard of track maintenance. The benefits derived, such as the reduced wear and tear on the rolling stock, the reduction of accidents which may result in deaths and injuries, and the reduction of haulage delays are not readily expressed in terms of dollars and cents. The experiences of other companies, both mining and railroading, serve as a guide in determining the degree of haulage maintenance which can be justified economically.

Despite a number of adverse conditions St. Joseph Lead Co. has steadily increased production. First, the orebodies become smaller, requiring a greater number of operating places. Secondly, the main haulage lines become longer year after year, and require a proportionally greater volume of materials and manpower for construction and maintenance. Thirdly, the shortage of materials and manpower during the war often resulted in the use of inferior products and sometimes the employment of an inferior type of man.

The above problems were met and solved in a number of ways. The large underground shops of the two general mines were well supplied with both machines and experienced men and not only corrected mechanical weaknesses of the rolling stock as they became evident, but also added a number of improvements in both structure and equipment. The shops were materially responsible for overcoming the mechanical weaknesses of the locomotives and also for improving their braking system, reducing the safety hazard on steep grades. The track gangs were able to reduce many grades and in certain cases to build new haulage lines to bypass the more critical areas. Nothing could be done to decrease the length of the haulage system appreciably. However, by further improving the control and signal systems,

it was possible to handle a greater tonnage per operating shift.

The Mine Locomotives: The Federal No. 17 underground shop overhauls all district locomotives, which range from 2 to 25 tons in size and consist of seven different makes, each make including two or more size ranges.

Locomotives of the 13-ton and heavier class are used on main line haulage; the smaller reel-type units are used for heading motors and for utility work. All locomotives, except the 25-ton G. E., are equipped with screw-type mechanical brakes and dynamic braking. While very dependable, the manually operated screw-type brake requires time and effort before it becomes effective. Therefore, when quick stops were necessary to avoid accidents, it was the common practice to reverse the motor as a means of braking. Recently, hydraulic systems have been installed on two 13-ton locomotives which appear satisfactory so far. This system has an operating valve which controls the brake-cylinder pressure, making it possible to obtain the degree of braking

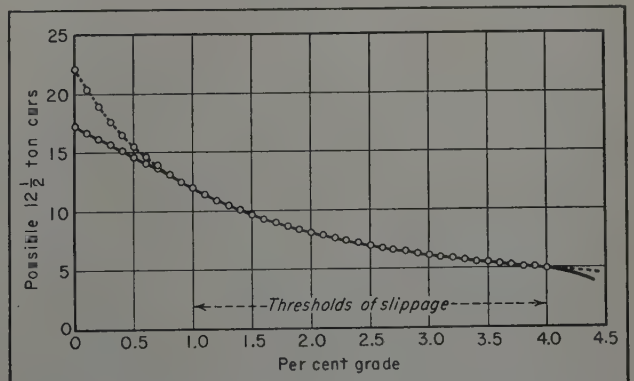


Fig. 5—A graph showing the number of 12.5-ton cars of ore which may be pulled by a 15-ton locomotive at various grades.



Fig. 6—Building the new type of 12.5-ton ore cars in the No. 17 underground shop. Cars have all-welded construction.

desired. An accumulator tank in the system is sufficient to brake the train in event of power failure.

The principle of dynamic braking was introduced to the district with outstanding success over 15 years ago. While this is effective only on a locomotive in motion, it is ideal for controlling the speed of the ore trains. This method of braking requires a special controller with a dynamic braking position which completes a circuit when applied, converting the motors of the locomotive to dc generators. The locomotive drivers turn the generator and generate power which is dissipated in the system. Naturally, the braking effect results from the energy required to turn the generators. Any break in the dynamic braking circuit immediately renders the whole system ineffective. Other than improvements on the wiring and materials, this system has not been changed from the beginning.

In 1937 a 25-ton G. E. locomotive with air brakes was purchased. This type of brake has proved to be very satisfactory.

A new type of brake shoe is now being adopted on St. Joseph's locomotives. The old shoe and clevis unit were integral and fitted rigidly on the brake beam. The new shoe and clevis unit comprises two separate parts which fit together with a hinge pin. This arrangement allows the new type of shoe to fit itself to the wheel and thus evenly distributes the wear. Also it is necessary to replace only the new shoe, which weighs about 50 pct less than the old type and represents a saving in material and labor.

Fifty-three gathering and main-line locomotives are now in operation. During the past several years, numerous tests and studies have been made including maintenance costs, determination of car factor and weights, power, average track-trolley conditions, grades, locomotive characteristics, and operating methods. Performance data covering several months were reduced to graphs and tables, the grade being used as the common abscissa and data concerning demand, cost, power, tons, and cars being used as the ordinates. This group of curves was fur-

nished to the supervisors to assist them in solving haulage problems in view of operating conditions. In these graphs, solid lines indicate actual values, and dotted lines show the theoretical. Actual values show loads which can be started and moved with the minimum of maintenance and operating costs. Each operating point on the graph is in the apex of a V curve of total costs vs number of cars or tons. This curve, see Fig. 5, seems to be nearly the same shape for all grades and operators. The point used is from a curve for average track and an average operator. Poor track or a careless locomotive engineer would displace this curve upward in its entirety, while excellent track and locomotive engineers would displace it downward. Extremes in either direction would move the curve to the right or to the left in addition to the vertical movement. However, these are not considered to be within the operating range. Thus if the system is properly designed, maintained, and controlled, the locomotive engineers only increase or decrease the costs without affecting the tonnage hauled to any great extent.

To use the graphs, the supervisor first determines the steepest significant grade for the run in question and reads the load from the curved line. On some parts of each run the locomotive is operated at its maximum capacity.

Mine Cars: Two types of standard mine cars are used in St. Joe mines today. The older type has a capacity of 48 cu ft, or about 2.6 tons of ore. It has a swivel coupling on each end, designed for use in the rotary dumps without uncoupling. In the Federal area these cars are used almost exclusively in development work behind the Conway loader. Leadwood, Desloge, and Bonne Terre still use more cars of this type than the other for hauling ore from the stopes. At present, there are about 1560 of these cars in service in the Leadwood-Desloge area, and about 1900 in service in the Federal area.

The new type of standard car, see Fig. 6, is about the length of three 2.6-ton mine cars, or 24 ft. It has a capacity of 220 cu ft, or about 12.5 tons of ore. It has the same kind of swivel coupling at each end so that mixed trains may be dumped in the rotary dump without uncoupling. At present, 620 of these large mine cars are in service. These cars have proved to be far more efficient in both loading and haulage operations than the older type. The last 100 large cars purchased have been equipped with roller bearings for test purposes.

The first group of the 12.5-ton mine cars was introduced in the Federal area about 1941 and tested over a long period of time. It was early apparent that despite the many problems that had to be worked out, the big car had many advantages over the 2.6-ton car. It occupied the space of three small cars but had the capacity of almost five, a great advantage in both switching and shovel loading, and much shorter trains could carry the same volume of ore on the main line track. The coil springs made easier riding cars, and this in turn greatly reduced the wear and tear on the track joints. Excessive mucking was eliminated by adopting a rounded bottom in the big car. It is now anticipated that the roller bearings will effect a savings in both lubrication and maintenance. Moreover, this type of bearing should have less starting friction.

A few of the A-dump type of cars are also in service for the purpose of utilizing waste rock to build fills for the track system. A small number of 2-ton mine cars for operation on the 30-in. gage

track are in service on the upper level of the National mine.

The rubber-tired shuttle car is making a very impressive showing in the Leadwood-Desloge area. Shuttle cars are also being used in the Doe Run and Mine La Motte areas. The new mines, Hayden Creek and Indian Creek, are being designed for trackless mining entirely. Although this paper is concerned with rail haulage, it must be kept in mind that in the stopes and headings a shuttle car has a number of advantages over the conventional mine car which requires an extensive track system. Although the shuttle car range of operation is still limited, haulage distance of the shuttle car trolley system has been extended to better than 2000 ft between the heading and the dumping point.

In the Mine La Motte area and the recently opened Doe Run mine, diesel trucks have been operating between the loader and ore bins. While their tonnage of ore is not very significant to date, this type of equipment appears ideal for open pit and those shallow mines having a truck roadway running underground.

Electrical Equipment and Installations

Power is delivered from the surface transformer stations underground through drill holes to the motor generator sets at 440 v ac. These in turn deliver power at 275 v dc to the trolley systems. The 440-v ac power is also run to the shovel transformers and operates the shovels at 220 v ac.

The Motor Generator Sets: The 45 motor generator sets supplying dc power to the trolley systems are rated between 100 and 300 kw. All generators are compounded, and most motors are synchronous with full automatic controls. Generator stations are semi-permanent installations, and are carefully constructed and maintained, see Fig. 7. Many operate continuously for both haulage power and for general power factor correction.

Regulation is obtained by a careful study of the location of the set. Four different areas are recognized in the setting regulation. If the set is at or near a central hoisting shaft, the location would be called *central*. If out a short distance on a main haulage line serving three mines or the equivalent, it would be considered as an *inner route* location. If further out on a main line and serving two mines, it would be considered an *outer route* location. If



Fig. 7—Two large motor generator sets with panel board located in the No. 17 underground shop. Sets of this type supply dc power at 275 v and 1800 amp.

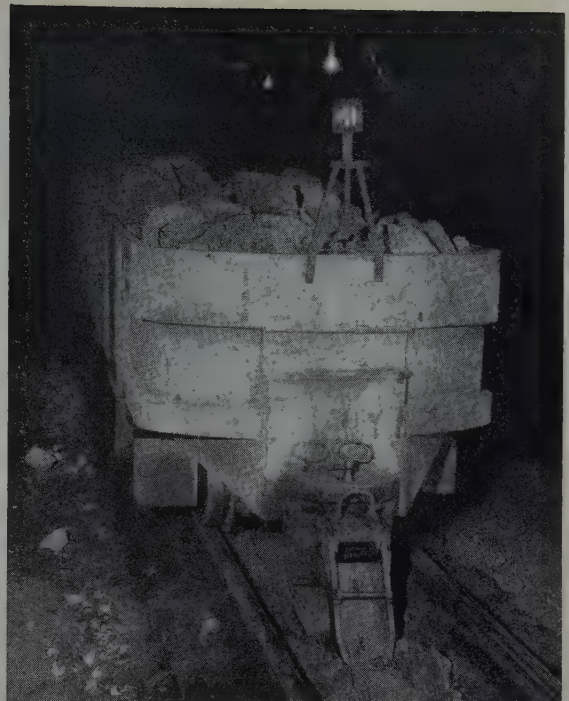


Fig. 8—The 12.5-ton ore car showing reflector tail light and hinge-type stubby trailer.

located in the outer zone serving only one mine, its location would be considered *fringe*.

If the central location has a regulation of V volts, then the following zone would be V plus 10, the next V plus 20, and the fringe V plus 30. This provides an increasingly stiff system, electrically, as the central area is approached. In turn, it permits the outer machines to retire from the load sufficiently to force adjoining sets to assist. This allows the outer set to stay on the line, increasing the power capacity available at a slightly lower voltage. Since haulage loads are transient in nature, the generator circuit breakers are set to open at a rather high current value. Normally, these ratings are set for between 75 and 100 pct overload based on the rating of the machine.

Trolley and Feeder Systems: The track and trolley-feeder systems have been built upon the twin standpoints of capacity and maintenance. Either welding or wedge bonding is used on the main line rail joints, with welding for the heavier loads. Weld joints show conductivities between 130 and 150 pct of that of the solid rail. All trolley is size 4/0, which has proved to have good wearing life and reasonable current-carrying capacity. Where heavier loads are to be handled, a feeder is added, sized in proportion to the load. Feeders are held in slings directly over the trolley for best usage of copper and trolley insulators.

Signal Systems: All traffic is controlled by some kind of signal. The general areas in the vicinity of the two central hoisting shafts are under the control of the respective general mines. All other traffic is controlled by a pull-type system of red and green signal lights initiated and released by the locomotive engineer. While subject to many operating disadvantages, these block signals still represent a practical solution of the problems of maintenance and safety. One central hoisting area uses manual signals controlled from a station at the rotary dump; the other uses an automatic block system.

In this latter system, the area is divided into a number of different sections or blocks. The presence

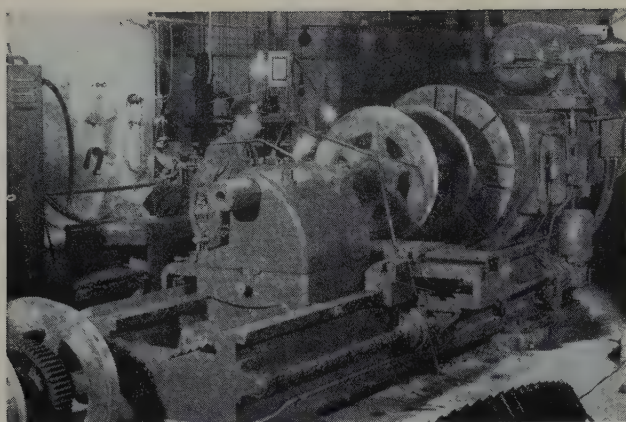


Fig. 9—A 48-in. engine lathe in the No. 17 underground shop machining the drivers for a 15-ton locomotive.

of rolling stock in any block shorts out the block track relay which operates the relays controlling the signal lights for that block and any priority connections which might be set up. In recent years, a semi-automatic system depending on contactors operated by the trolley pole collector shoe has been used to a limited extent.

A few main line locomotives are equipped with radio-telephones for two-way communication. They are single-frequency transceivers powered from the trolley. These sets are used to expedite supplies and facilitate operations in general. They are not a substitute for the signal systems. Areas of about 1-mile radius require little electrical connection. On main haulageways with limits extending about 10 miles, special circuit provisions must be made. A special antenna is installed above the trolley and selectively coupled to it. Booster stations are installed where required. Because of the volume of messages, two frequencies are in use. These cover separate areas which contact only at a central hoisting shaft.

No signal system is any more effective than the enforcement of the regulations governing its usage. In the interests of maximum clarity, operating rules are kept to a minimum. These are based primarily on safety and secondarily on the most economical use of the equipment. Enforcement is implemented by emphasis on cooperation and elimination of hazards rather than by punitive action. All movements are governed by the signal systems with prior arrival the only criterion except in the automatic block area previously mentioned. General methods of operating the equipment are taught during the training period. Then the locomotive engineers of each mine group set up their own operating schedule and procedures subject to the approval of the mine captain.

It may be stated that the chief purpose of the St. Joseph's signal system is the protection of the men. The automatic block signals, the pull-type light signals, the sirens on the locomotives, and even the radio-telephones are designed to warn personnel of the approach or location of a locomotive. The reflector type of tail light is designed to assure the locomotive engineer that his train is still intact and would also be a visible warning to any other locomotive engineer that a loose ore car was on the track ahead of him. The new hinged-type stubby trailer is designed to prevent an ore car from rolling backward. It is strong enough to stop any train and will either derail the cars or plow up ties until it brings the train to a halt, see Fig. 8. A safe haulage system is an efficient one.

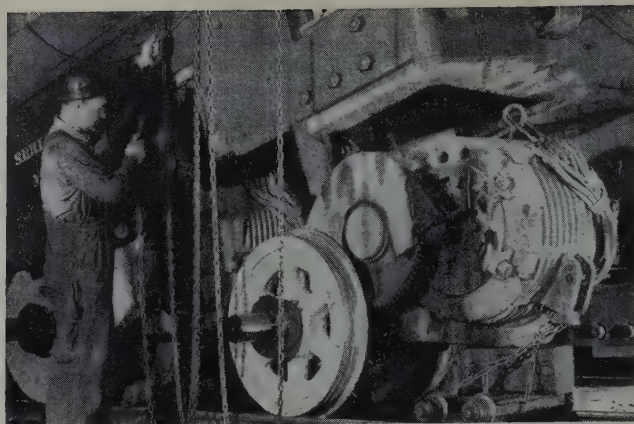


Fig. 10—Removing the motors from a 15-ton Goodman locomotive on one of the No. 17 locomotive pits.

Maintenance facilities have had to keep pace with the expanding haulage system. These include repair, overhaul, and redesign, see Fig. 9. This has made it possible repeatedly to adapt existing equipment to meet new demands. Standardization of parts has helped to reduce stock and provide sufficient spares where needed. Armoring and the relocation of controls and connections has not only resulted in improved service and safety conditions, but has also greatly increased the usable rating of the equipment. For example, improvements of the 13-ton class of mine locomotives have increased their capabilities until they not only can pull at least 30 pct more load, but also can stand far more rough service. It is evident that elaborate shop facilities are required to overhaul and to build mine locomotives, see Fig. 10, and other types of heavy mine machinery like the St. Joe shovel. Because of the size of the shafts in this district and the increasing size of mechanical equipment, it was long ago decided to establish the major mine repair shops underground. While each mine group has a small shop for minor assemblies and repairs, the major overhaul, repair, and rebuilding jobs are performed at either the Federal No. 17 underground shop or the Leadwood No. 12 underground shop.

No. 17 Federal Underground Machine Shop: Unlike a surface machine shop which must be housed in a building, this underground shop has almost unlimited space in the old mined-out areas. The average height of the ground is over 15 ft. Since space is not at a premium, this shop continues to spread out. Because the entire area is covered with a maze of track, the handling of supplies does not present a problem. This shop covers over three acres of area, and its normal crew consists of 96 assorted machinists, repairmen, welders, blacksmiths, and electricians. It operates on day shift only. On night shift a few repairmen are always on hand for emergency service.

Acknowledgments

The author gratefully acknowledges the assistance of Walter D. Duffy, Jr., Mine Captain of Federal No. 17; Bert L. Beal, Jr., Electrical Engineer; John J. Northcutt, Mine Engineer; and Theodore O. Seiberling, Bonus and Contract Engineer, in gathering the data for this report. Vincent D. Aubuchon, Assistant Safety Inspector, deserves credit for the photography. Many other members of the staff of the St. Joseph Lead Co. were most helpful with their comments and suggestions.

Earth Resistivity in Groundwater Studies In Illinois

by Merlyn B. Buhle

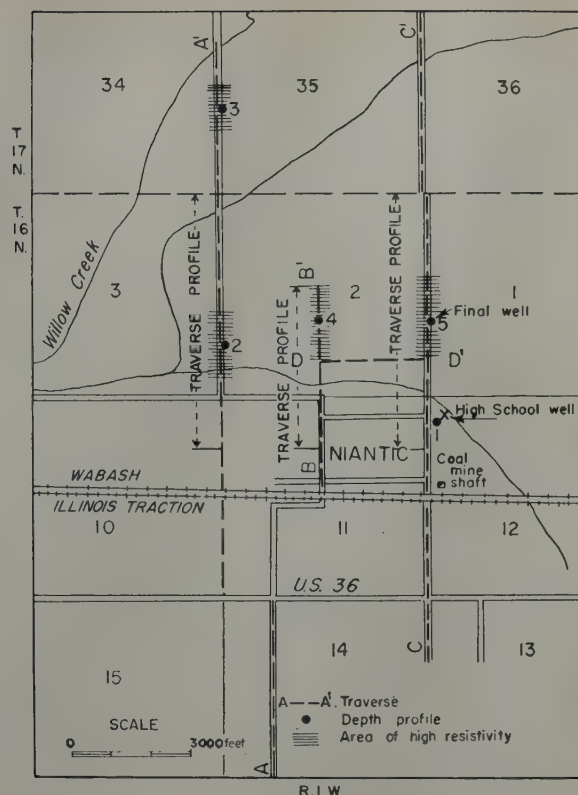


Fig. 1—Map of Niantic area, Macon County, Ill., showing resistivity traverses and areas of high resistivity.

The value of the earth resistivity method in groundwater studies in Illinois has long been recognized. Owing to the sharp electrical contrasts between sand and gravel deposits and glacial till; alluvial silt or underlying shale, much of the earth resistivity prospecting has been of a relatively elementary nature and has been highly successful. Approximately 90 pct of earth resistivity interpretations by the State Geological Survey have been accurate for practical purposes. Four studies involving fairly simple geologic situations are presented in this paper as typical cases.

FOR the past 20 years electrical earth resistivity exploration has been used in Illinois in many phases of study undertaken by the State Geological Survey, chiefly in locating and outlining deposits of water-bearing sand and gravel from which to obtain ground water for municipal and industrial supplies. At least 43 municipalities in the state derive their total water supply from deposits of sand and gravel discovered by resistivity methods. These deposits occur in both Quaternary and preglacial valleys and within much of the glacial drift which covers a large portion of the state.

In recent years there has been much demand from rapidly growing municipalities for help from

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Discussion on this paper, TP 3496L, may be sent (2 copies) to AIME before June 30, 1953. Manuscript, Feb. 20, 1952. Revised Dec. 8, 1952. New York Meeting, February 1952. This paper is published by permission of the Chief of the Illinois State Geological Survey.

the Illinois Geological Survey to extend existing well fields. Water supplies for many and varied industries constitute a considerable portion of the Survey's studies. The diminishing need of water supplies for railroads has been accompanied by an increasing demand by the oil industry for water for secondary recovery programs.

The instruments used by the Illinois Geological Survey follow closely the commutated direct current circuits developed by O. H. Gish and W. J. Rooney¹ using the Wenner² electrode configuration. More recently replacement of the hand-driven commutator by a synchronous vibrator has led to increased ease of operation and accuracy. Continuing improvement of equipment and technique has led to many changes in the instruments, but the operating theory remains the same.

In the field operation at a selected station, four electrodes are set at equal distances in the ground 6 to 10 in. deep, in a straight line. A 16 to 20-cycle current is applied to the two outside steel electrodes

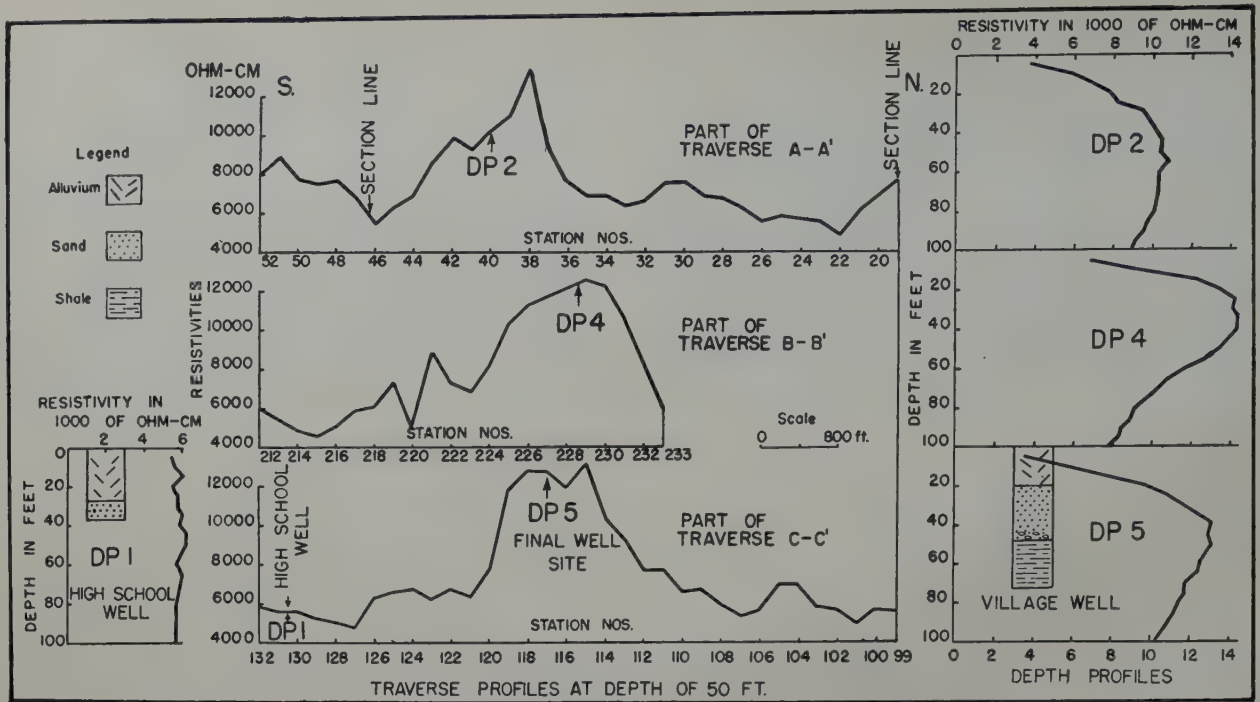


Fig. 2—Resistivity traverse profiles, depth profiles, and well logs in Niantic area.

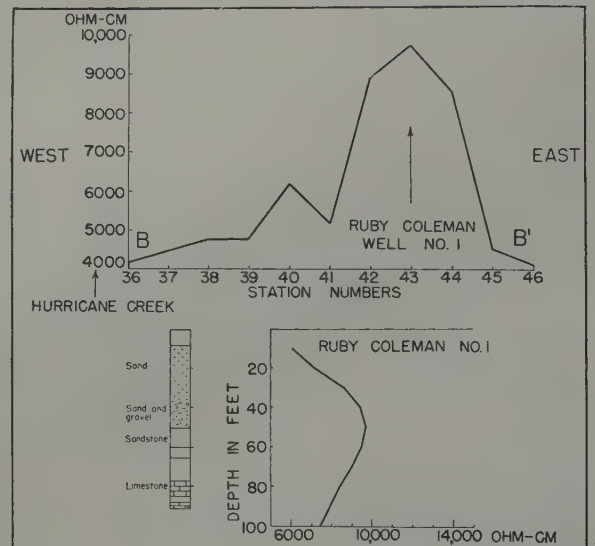
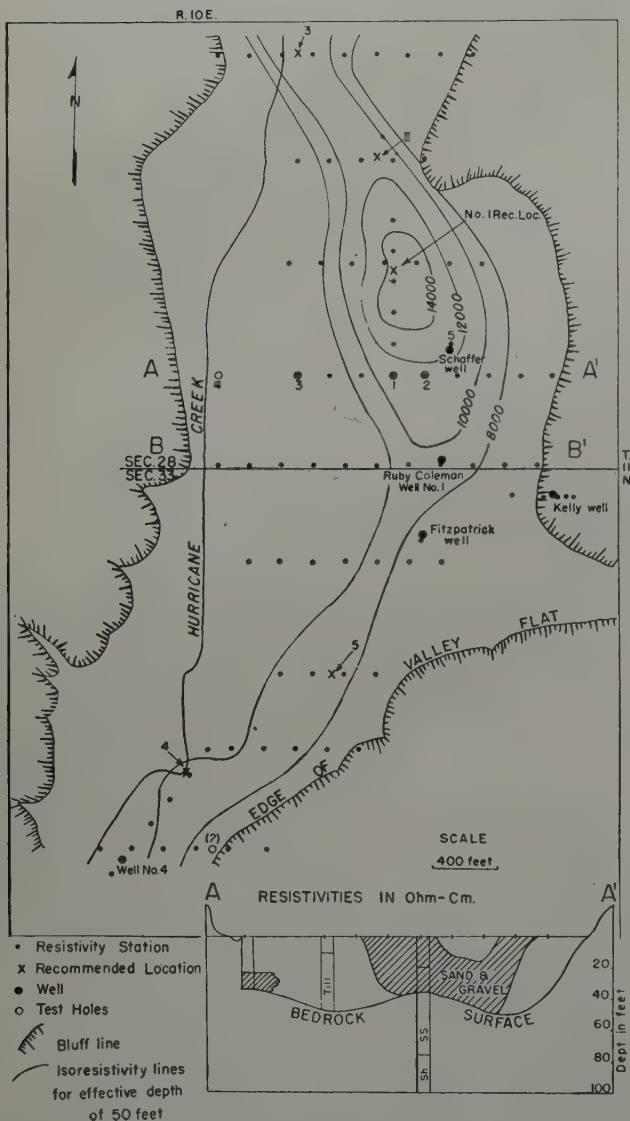


Fig. 3 (above)—Map of Hurricane Creek Flat, Cumberland County, Ill., showing location of resistivity stations and contours through points of equal resistivity at 50-ft electrode separation.

Fig. 4 (left)—Resistivity traverse profile BB', depth profile, and well log on Hurricane Creek Flat.

Fig. 5 (opposite page)—Resistivity traverse profile AA', depth profiles, and well logs on Hurricane Creek Flat.

and a measurement is made of the drop in electrical potential between the two inside copper electrodes. The apparent resistivity is read directly in ohm-cm on a specially calibrated potentiometer. The spacing of electrodes is nearly but not exactly equal to the depth of the earth materials affecting the resistivity. Some geologic situations are well suited to the approach by the step traverse method, by which a single reading at a single electrode spacing is made at each station, while others, more complicated, require depth profiles from a series of readings at each station at different electrode spacings.

For the groundwater work in Illinois, readings are not usually taken to depths much below the surface of the bedrock, since only the glacial drift is usually under investigation. In areas where a mathematical approach to the solution of the problem may be needed, readings are taken to at least twice the estimated depth to bedrock. In areas where the glacial drift is thick, and where a large thick deposit of water-bearing material is sought, stations are usually spaced 1/10 to 1/4 miles apart. In other areas, such as flood plains of streams where elongated deposits of sand and gravel are likely to have abrupt sides, stations are frequently located no more than 100 ft apart.

Approximately 90 pct of the resistivity surveys for water supplies have been made in a 200-mile wide belt running east-west across central Illinois. For the most part this area is underlain by Pennsylvanian rocks, much of which is shale in the upper part. Because shale is usually of lower resistivity than sand and gravel deposits in the glacial drift, an elementary type of resistivity prospecting has been highly successful. Test drilling of recommended sites has proved the interpretation of re-

sistivity measurements to have been correct in about 92 pct of the cases. The methods using type curves and a mathematical approach have been used experimentally in Illinois in fewer than a dozen instances.

Four specific resistivity studies are presented to illustrate the type of investigation made by the Illinois Geological Survey. Two are studies in which interpretation of the data proved correct, and two are studies in which the interpretation was wrong to some extent.

Location of the first study is Niantic, Macon County, in the east-central part of Illinois, on the surface of the Illinoian glacial drift, west of the edge of a moraine of Wisconsin age. The area shown in Fig. 1 is relatively flat and there is no surficial evidence of sand or gravel deposit. A coal mine shaft at the edge of the village leads to extensive workings beneath the village and surrounding area. It was reported to be a dry mine.

Parts of three resistivity traverses are shown, AA', BB', and CC', extending south to north across the area with stations 200 ft apart. Electrode spacings of 25, 50, and 75 ft were used at each station except at points 1, 2, 3, 4, and 5 where readings were taken at 5-ft intervals to a maximum spacing of 100 ft. The areas of high resistivity as well as the locations of the depth profiles are indicated on the map.

Fig. 2 shows the pertinent data plotted. The conspicuous feature on each of these traverses made at the 50-ft electrode spacing is the localization of the resistivities above 8000 ohm-cm. The geologic conditions found by test drilling at depth profile 5 can be expected to occur also at sites of profiles 2 and 4 where similar curves were obtained. The low flat depth profile near the High School well indicated very little permeable material, which agrees with the log record of the well. A producing well was drilled at the location of depth profile 5 on the basis of the resistivity data. It had a specific capacity of 64 gal per min per ft of drawdown.

Fig. 3 presents the results of a second study, located 6 miles northwest of Casey, Cumberland County, in the southeastern part of Illinois, on the flood plain of Hurricane Creek. In Pleistocene time, a stream flowed in this valley from northeast to

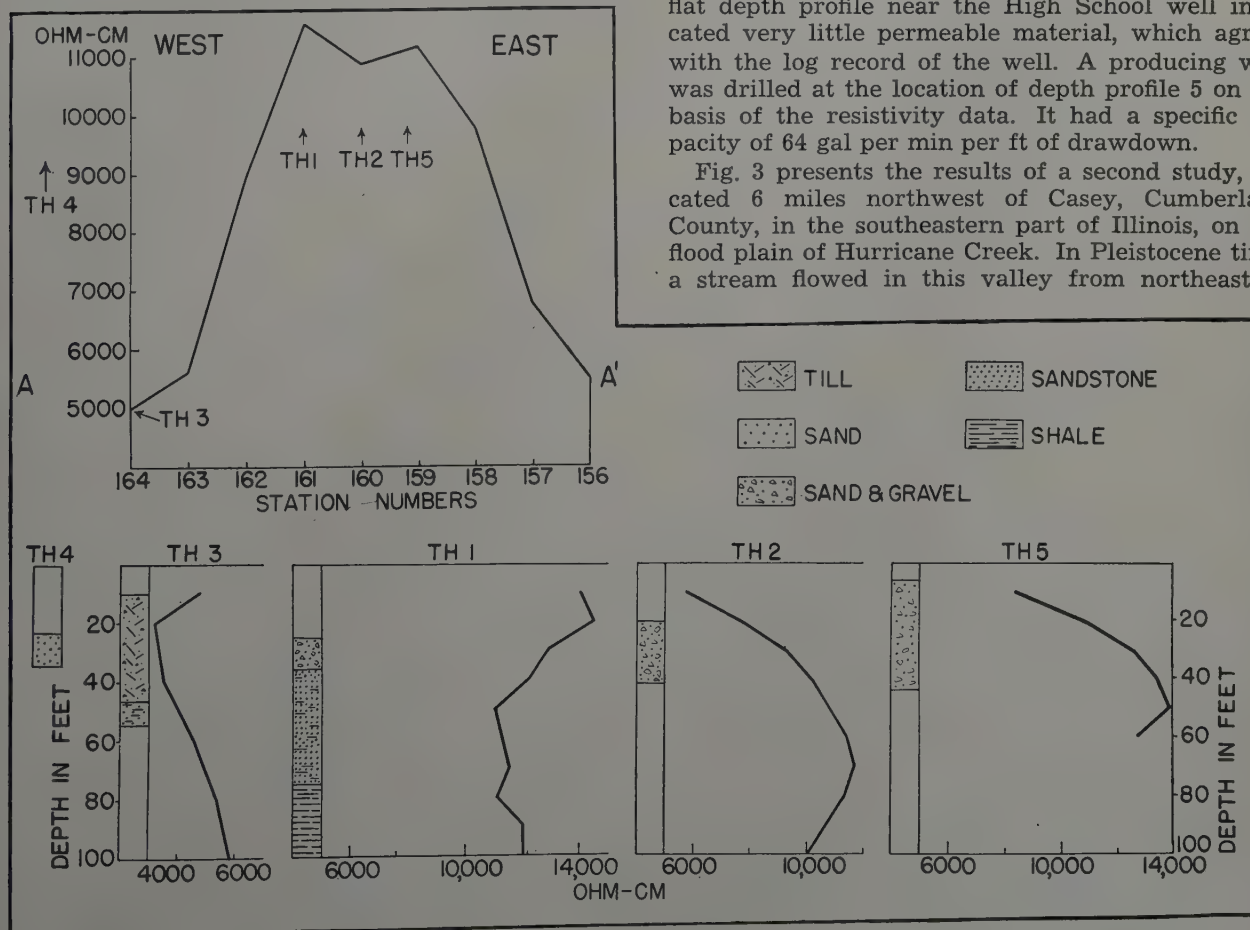


Fig. 4 presents resistivities obtained at the 50-ft electrode separation along traverse BB', the resistivity depth profile at the Ruby Coleman well No. 1 site, and the log of test drilling at the site.

Fig. 5 presents the results along traverse AA', taken also at the 50-ft electrode separation, together with numerous test holes drilled along it, and the depth profiles and logs of these test holes. It will be noted that test holes 1, 2, and 5 on traverse AA' and the Ruby Coleman well No. 1 on traverse BB', which were drilled on the resistivity highs, encountered thick sand sections, whereas test holes 3 and 4, which were drilled off the resistivity high, encountered little or no sand. The curves obtained on and off the deposit and resistivity high are vastly different. Additional resistivity measurements were made in the fall of 1951. On the basis of additional information and results of previous test drilling on resistivity highs, recommendations were made for a test drilling program for spring, 1952.

The Ruby Coleman No. 1 and Fitzpatrick wells have been abandoned; well No. 4 was a slotted pipe installation of very low capacity and is rarely pumped. The Kelly well is perched on the valley wall at the edge of a gravel pit. Only the Schaffer and Kelly wells are now in continuous production.

Fig. 6 shows the results of an extensive survey completed a few years ago 2 miles northwest of LaHarpe, Hancock County, near the western border of Illinois. In this region the Pennsylvanian rocks become thin or pinched out, so the bedrock surface may consist of shale or limestone of Mississippian age. In the valley bottom of Crooked Creek a resistivity high was mapped. The valley flat itself, which is everywhere at least ¼-mile wide and lies about 50 ft below the general upland level, is a typically favorable location for a sand and gravel deposit. Some sand and gravel bars are exposed in the creek meanders. Sand at least 10 ft thick is reported to have been encountered when piling was sunk for the railroad bridge at the creek crossing. Few records of wells on the uplands are to be

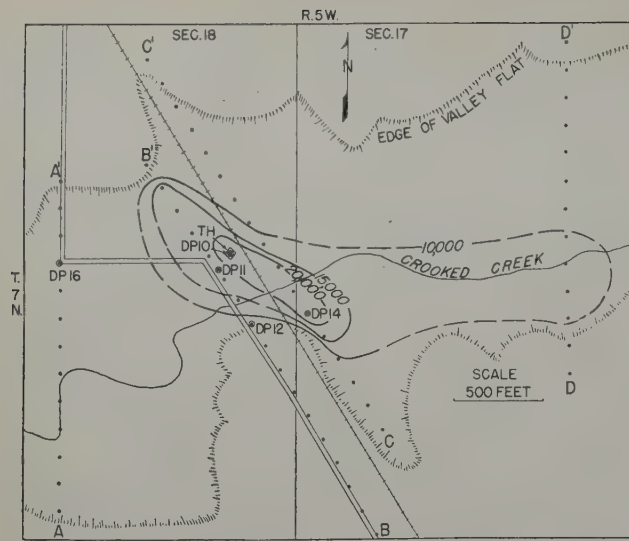


Fig. 6—Map of Crooked Creek Flat, Hancock County, Ill., showing locations of resistivity stations and contours through points of equal resistivity at 100-ft electrode separation.

southwest along the frontal moraine of the Wisconsin ice. The valley contains considerable sand and gravel deposited by the glacial water, but the location of clean portions that are good aquifers is generally unpredictable.

Measurements at all stations on traverses AA' and BB' were taken at the surface to an effective depth of 120 ft by 10-ft intervals. This was done before test holes in the flat revealed no depths to bedrock greater than 50 ft. Later station measurements were taken only to an effective depth of 60 ft. Traverse BB' was run first, and a test hole completed as a producing well, known as the Ruby Coleman No. 1 well, was drilled at a site chosen from the resistivity data. Lines of equal resistivity were drawn as shown by Fig. 3.

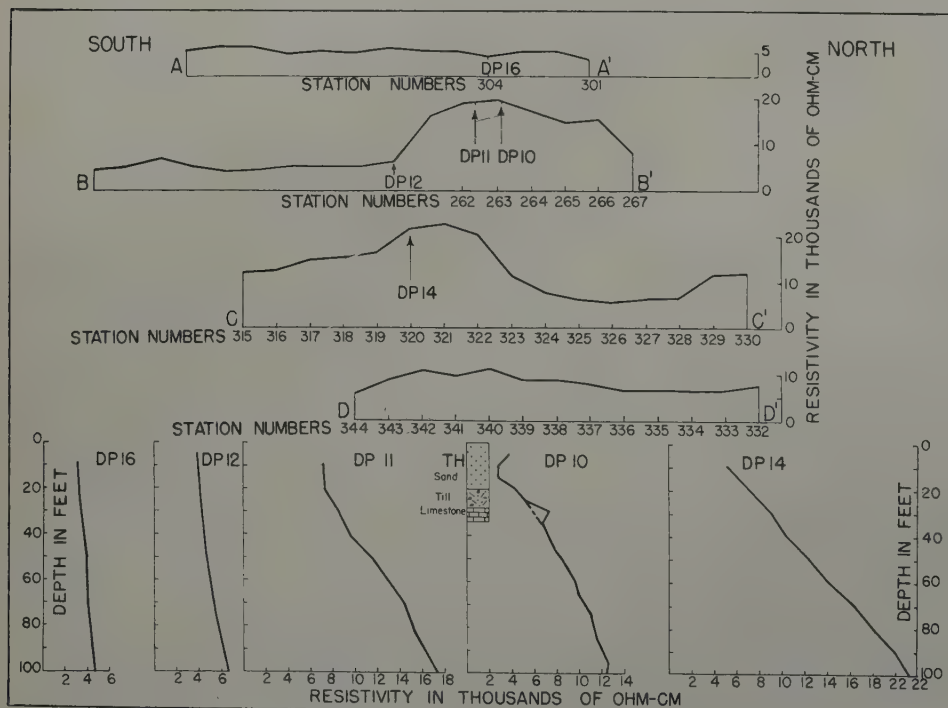
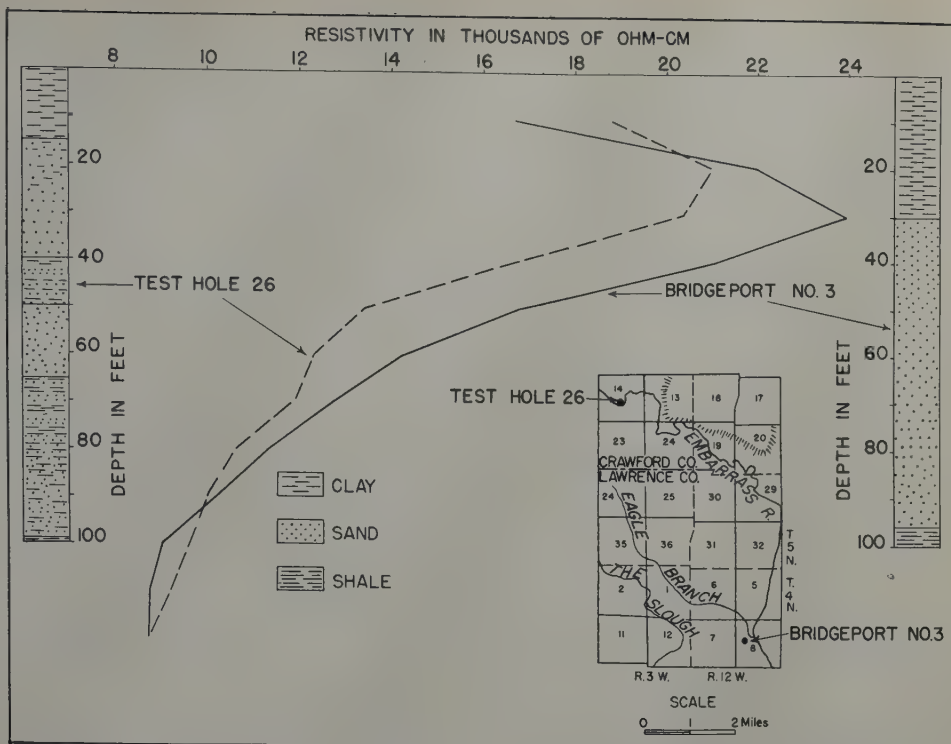


Fig. 7—Resistivity traverse profiles, depth profiles, and well log on Crooked Creek Flat.

Fig. 8—Part of Embarrass River Flat, Lawrence and Crawford Counties, Illinois, showing resistivity depth profiles, logs of wells, and locations of depth profiles and wells.



found, but those available indicate that the glacial drift may exceed 80 ft in thickness. Four traverses of this survey are shown crossing the flat and extending to the upland on either side. An electrode spacing of 100 ft was used and the stations were spaced at 150-ft intervals. Five depth profiles were obtained, extending from the surface to an effective depth of 100 ft at 5-ft intervals. Lines of equal resistivity were drawn on the basis of these data.

On Fig. 7, traverse AA' shows no resistivity above 5000 ohm-cm, but the other three traverses indicate highly resistant material forming a typical sand and gravel pattern in the creek flat. Depth profiles 16 and 12 were obviously off the resistivity high, whereas depth profiles 11, 10, and 14 are just as obviously on the resistivity high. A test hole, TH, was recommended at the site of depth profile 10. It disclosed 20 ft of sand immediately below the surface, followed by 8 ft of alluvium, which in turn rested on limestone at least 6 ft thick. No well was made at this site because it was believed that there would be insufficient water for the project and because it was obvious that the resistivity pattern was caused by this near-surface limestone.

Fig. 8 presents the results in an area traversed by the Embarrass River in the region lying south of Robinson and Oblong in Crawford and Lawrence counties. The flood plain of this river is many miles wide and relatively flat. In this area the river follows a preglacial channel and is tributary to the Wabash River, which also follows a preglacial channel. The sites of both resistivity stations are on the same river flat, near the Embarrass River itself. The underlying bedrock at both sites is Pennsylvanian shale of low resistivity. It may be inferred that the geologic conditions at both sites are the same.

The Bridgeport No. 3 well had been completed as a Kelly well in a solid sand section 66½ ft thick and produced 1200 gal of water per min with a very slight drawdown. Its production was to be used

nearby for water flooding in an oil field. A resistivity curve was obtained close to Bridgeport No. 3 well to check a known good sand section. Later a curve was obtained in the northwest corner of the area shown in Fig. 8 at the site of Test Hole 26 so similar to the curve obtained at Bridgeport No. 3 that drilling seemed warranted.

When Test Hole No. 26 was drilled, however, it was found that although there was a very favorable sand section, some clean and highly permeable, the overall picture indicated low total permeability because of many thin clay laminations and possibly crossbedding. From the fluid-loss record it was estimated that a completed well would produce about 300 gal of water per min. As this amount was inadequate, though a production of 300 gal per min would be very acceptable for many purposes, a well was not drilled.

Another test hole, drilled on the Embarrass River flood plain on the basis of a resistivity curve similar to those in Fig. 8, encountered a thick section of clean sand, but there were in addition so many finely divided particles of wood as to preclude the possibility of developing a high production well. Unfortunately it has not been possible to detect the presence or absence of wood in a sand and gravel by resistivity methods.

These studies indicate that resistivity patterns typical for sand and gravel deposits do not always assure that sand and gravel actually occur at depth. However, the percentage of correct interpretations in these groundwater studies is very high. It must be emphasized that a knowledge of local geologic conditions is essential to interpretation of resistivity data.

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¹O. H. Gish and W. J. Rooney: Measurement of Resistivity of Large Masses of Undisturbed Earth. *Terrestrial Magnetism* (1925) **30**, pp. 161-188.

²Frank Wenner: A Method of Measuring Resistivity in the Earth. *Bull. U. S. Bur. Standards* (1916) **12**, pp. 469-478.

physical survey be made to confirm the reported electrical indications, especially with methods not used by the German geophysicists. This view was endorsed by Mr. Kerr-Cross, mining engineer of the Geological Survey of India. Eventually, from December 1949 to May 1950, a field party led by the present writer made a re-survey of the area, employing self-potential, resistivity, and magnetic methods. A comprehensive account of this survey is shortly being published by the Geological Survey of India. The present paper deals only with some aspects of the self-potential anomalies observed in the locality.

A systematic grid was laid out over an area of 1.3 sq miles, where the old trial pits and shafts exist, as well as the isolated sections examined by Messrs. Piepmeyer & Co. The survey layout is shown on the opposite page. Note that a *tank* is a small reservoir, a *tank bund* the reservoir embankment.

In a considerable part of the area electrical resistivity measurements revealed a highly conductive formation which on closer examination was found to consist of the wet clays of the bed of a small reservoir. These clays occur as superficial deposits, either exposed on the surface or covered up by thin layers of sand. The resistivity of the wet clay is as low as 0.7 meter ohms, whereas resistivity of the adjoining ground, composed of sandy soil and more or less decomposed granitic gneisses, is from 20 to 30 meter ohms. The strong indications found by Piepmeyer & Co. were, in fact, all caused by these highly conductive clays in the tank bed. The inductive method they had employed for their survey must have been extremely sensitive to superficial conductors. The self-potential measurements made by the second field party also revealed no indications which could be ascribed to the existence of any sulphide orebody in depth. It was therefore concluded that no useful purpose could be served by trying to do any further drilling, shafting, or such other costly exploratory work in that area.

During the course of the surveys, self-potentials of +40 to 50 mv strong were noted in the area. These anomalies, which had no relation to the ex-

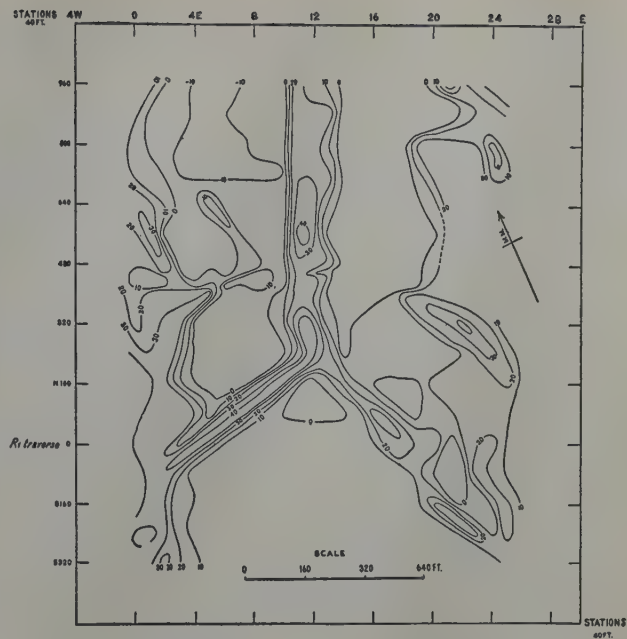


Fig. 2—Equipotential (self-potential) plan, subsidiary grid, Garimenapenta. S.P. contour intervals 10 mv.

istence of orebodies, were chiefly caused by sub-surface movement of ground water in the sands along the beds of narrow creeks in the area and partly by the natural potential difference existing at contact of wet clays and sands. Since this feature was considered to be of some general interest, extensive observations were made in the field and later at headquarters in a series of model experiments to study the nature of these self-potential centers. This paper records the results of such extended observations.

Self-potential Anomalies Noted in the Main Grid-Layout

Fig. 1 shows the numerous self-potential profiles obtained over the whole area. The potential difference between the primary base station, kept at 0 station of S 400 profile, and the temporary base point of each line were determined and checked periodically during the surveys to note variations. Fluctuations did not exceed 5 mv. The potential readings were adjusted to the primary base station, the value of which was assumed to be zero.

The feature at once remarkable in all these profiles is the absence of any distinctive negative potentials. A number of zones of positive potential do occur, and the manner in which these narrow zones delineate some prominent gully, or *nullah*, courses is fairly obvious. The two principal zones start from stations 20 W and 110 E, on profile S 4400, and go with N.N.E. and N.N.W. trends respectively. A third zone starting from station 40 E on profile S 2200 goes northward and joins the eastern zone near station 50 on 0 profile. Between profiles S 1600 and 0, especially in the portion coming within stations 10 E to 70 E, there is a swarm of positive potentials on the sands. Further north, after profile N 400, the traverse lines were all entirely in the small reservoir into which water had gone, leaving only the dry silt and clay exposed on the surface.

It may also be noted from the plan of the survey layout that the traverses S 2600, S 3200, and S 3400 cut across some of the most prominent old

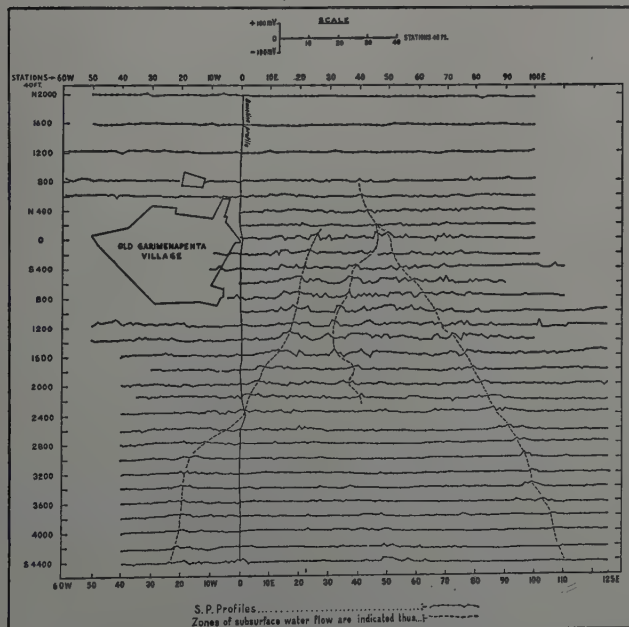


Fig. 1—Self-potential profiles on the grid lines, Garimenapenta.

workings where the pegmatites had shown copper ores, chiefly malachite. The S.P. profile, however, shows no significant indications in association with any of those old workings. In fact, within the whole area there is no indication of any negative center which could be ascribed to the existence of any mineralized formations or orebody. The only anomalies are the positive potential zones, not more than 50 mv strong, and these are in conformity with the trends of the nullah courses in the area.

Intensive Measurements in the Zones of Positive Potentials

A series of closely spaced lines were laid out in a small portion of the area where the positive potential zones appeared as a pair with a low between them. This small portion is denoted by the short subsidiary lines between main traverses S 600 and S 2200, confined to stations 10 E to 60 E generally, as shown in the layout. S.P. readings were taken on 250 stations and equipotential lines drawn by interpolation. The resulting plan is given in Fig. 2. From this plan it is evident that the positive potentials show definite trends, tracing with remarkable fidelity the pattern of the nullah courses. The way in which the streams at the S.W. and S.E. part of this layout coalesce into one stream at about station 12 on N 160 profile is unmistakably contoured by the strong trends in the equipotential lines.

Other features deserve comment. The streams are only generally indicated, but the magnitude of the potentials are not precisely in conformity with the directions of the flow of water. There are also many minor positive centers scattered here and there which bear no particular relation to the known stream lines.

In the field it was noted that a very thin layer of sand occurs on the surface, while clays occur beneath the sand. The stream beds are marked by a greater thickness of sands, some 2 to 4 ft thick, which are saturated with water and have a resistivity of about 4 meter ohms. The sands in such places are filling up a deeper channel which previously must have been carved out in the clays of the area. In most of the places where the positive self-potential peaks were noted there was no water flowing on the surface, but below a few inches to a foot the ground was saturated with water, and there were distinct trends suggesting lines of subsurface flowage. Several shallow pits were dug out and the potential differences between the clay and sand were measured. Readings taken in the sands gave always a higher value with reference to the base pot embedded in the clay. When the sands were removed and the clay occurring underneath, in depth, was contacted, nearly all the positive potential anomalies disappeared. Furthermore, readings taken after water was bailed out from the pits showed that the flowage of water caused a higher potential opposite the direction in which the water flowed. The sands are fairly coarse and highly permeable. The clay bed at the bottom acts as an impervious layer and the water in the sand flows along a very gentle gradient, in conformity with the channel formed in the clay bed. The free movement of water in the sands under a slight hydrostatic pressure has apparently produced an appreciable electrical potential anomaly.

In a summary of results of the detailed self-potential measurements carried out in the field, two points are important: 1—Between the clay and

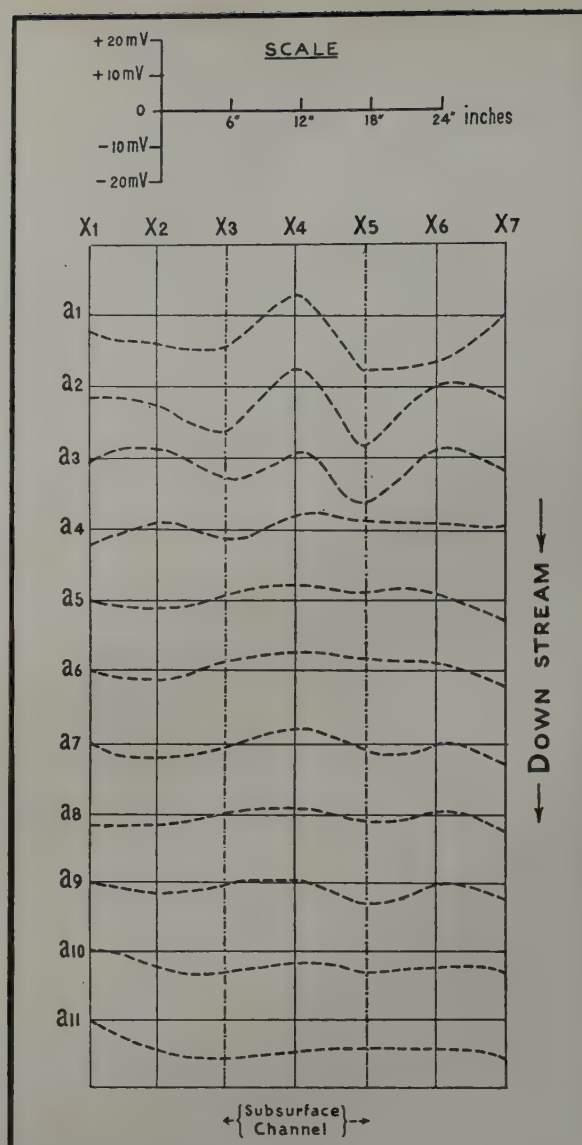


Fig. 3—Self-potential profiles observed in model experiment.

sands there is a more or less constant electrical relationship, the sands possessing always a higher potential. The magnitude of such potential anomaly is variable but small, not exceeding 20 mv. 2—When the water in the sands, occurring over the clays, has a marked tendency to flow, this potential difference is accentuated, the magnitude going up to 50 mv. The effect of streaming or electrofiltration potentials then becomes preponderant.

Self-potential anomalies of minor amounts are probably caused by other factors connected with the variations in permeability of the sands and electrical resistivity of the water. These latter factors, however, were not studied, but variations did not appear to be large in the small portion of the area where the intensive self-potential measurements were carried out.

Model Experiments

A series of experiments were carried out in the laboratory at Calcutta by Mr. S. C. Nandi to investigate the nature of the potential anomalies arising from a subsurface flow of water in sand over a clay bed. A large box of wood, 6 ft long, with waterproof lining, was used as a model tank. To simulate the conditions prevailing in Garimenapenta area, the tank was filled with a quantity of fine alluvial

clay covered over uniformly by sand. The thickness and shape of this clay bed were varied for different experiments. In all instances, the clay bed was covered with loose sand, finishing the top surface to a uniform plain level. Water was let in at one end of the tank by means of a controlled fine spray of fresh water from the tap. The subsurface flow of water in the sand depended upon the section and slope of the sandy bed. A suitable outlet for the water at the other end of the tank was also made, although the outlet could not be regulated as nicely as the inlet by fine spraying. On the whole, conditions of experiments simulated the field conditions.

Self-potential measurements carried out in the first series of experiments, with the clay bed having only a plain sloping surface and the sand filled over this clay bed, showed that where the thickness of the sand was small, the potential differences were only +2 or 3 mv, whereas in the portions where the sand was considerably thicker, the readings obtained were +7.0 mv, but anomalies were irregular.

In the second series of experiments, the clay bed had a channel 2 ft wide, and a gentle gradient. The whole of the clay was covered up by sand as in the previous case. With a spray of water giving a constant flow, the potentials measured along profiles, across the channel, distinctly showed a positive reading, but the magnitude was weak and the potential gradient very poor.

In the third series of experiments, a sharp channel, 1 ft wide, with steeper slope, was carved in the clay bed. This was filled with sand, and as usual a constant inlet of water was regulated at one end of the model tank. The self-potential profile obtained is given in Fig. 3. It will be seen, therefore, that in the central portion corresponding to the channel position in the curves a_1 , a_2 , a_3 , a very distinct and characteristic positive anomaly of +20 mv is outlined, faithfully denoting the trend of the channel in which there is a subsurface flow of water. This anomaly in the other profiles is not distinct or readily observable because they happened to be located in the lower reaches of the channel closer to the outlet, where stagnation of water rendered the potential anomaly very weak and indistinct.

Some further experiments were also tried to study the effect of movement of water under different gradients, but no important quantitative results could be achieved, owing to the limitations imposed by the size of the tank and the arrangement for inlet and outlet of water which could not be controlled over a wider range.

Compared to the magnitude of the self-potential anomalies noted at Garimenapenta, the experimental results showed about only half the value. In a qualitative way, however, the model tank experiments showed that a subsurface movement of water flowing through a sand bed gives rise to positive S.P. anomalies, outlining the channel bed along which the water moves.

In the self-potential measurements for well logging, the effects of electro-filtration potentials have been studied in great detail. According to Schlumberger and Leonardon,³ the magnitude of the *EMF* produced by filtration may be expressed by the equation

$$E = \frac{mRP}{V}$$

where *E* equals electromotive force, *m* a constant depending on the nature of the porous medium, *R*

the electrical resistivity, *P* the pressure, and *V* the viscosity of the flowing liquid.

It is obvious, therefore, that if the other factors are equal, *E* would vary directly with the differential hydrostatic pressure of the liquid. The sign of the *EMF* would be such that the electric current it causes to flow would possess the same direction as the liquid in movement.

In well logging the electro-filtration potentials are noted to be a minimum at the boundaries of the porous zone and a maximum in the most permeable section. The magnitude of the electrofiltration potential differences may be of the order of 100 to 200 mv, over a length of a few meters.⁴ If the formation is discharging water into the well, the potential anomaly will be positive with respect to an electrode kept at the base station on the surface. On the same analogy, subsurface flow of water in sand under hydrostatic pressure would give rise to potential differences and the anomalies would have a positive sign with respect to points where the hydrostatic pressure is 0. Leonardon⁵ has reported that water flowing in the sand of a sea beach would set up self-potentials due to electro-filtration and that water rising by capillarity in a formation would also cause a difference of potential.

Parke A. Dickey⁶ has found that natural potentials, as distinguished from electro-filtration and electro-chemical, are observable across the contact of sandstones and shales. He has put forward evidence to show that the potential difference is a specific property of the rock and its contained electrolyte. The cause of these potentials is surmised to be in the relative polar adsorptive capacities of quartz and clay particles.

In Garimenapenta, too, it was observed that the potentials measured on the sandy layers were always positive with respect to clays, but not more than 10 to 20 mv strong, unless of course, the sands were of the nullah bed where there was a subsurface water flow which greatly enhanced the effect, raising the difference up to +50 mv. The S.P. anomalies observed in the Garimenapenta area are therefore to be attributed to a combination of both aspects, such as 1—the natural potential bias, or the existence of an electrical double-layer between sand and clay, and 2—streaming or electro-filtration potentials due to subsurface flow of water.

Acknowledgments

The author would like to express his gratitude to Messrs. M. A. Alwar, S. C. Nandi, and M. L. Chatterjee for their valuable assistance in the field and in the laboratory. Thanks are also due to Mr. Haren Dutta for drawing the illustrations presented in this paper.

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Stream Pollution By Coal Mine Wastes

by Henry F. Hebley

This paper brings within the compass of one comparatively brief article a general description of the situation concerning the nation's water resources. It touches upon the phenomenal growth in the demand for water supply and emphasizes the problems facing the coal industry both with regard to acid mine water drainage discharged from active and abandoned mines and the suspended solids discharged to the stream system from wet coal preparation plants.

IT must always be remembered, with regard to the problem of water-borne industrial wastes, that stream pollution from any cause is just one factor in the comprehensive problem of water supply in modern times. Industries nonexistent 25 years ago are now flourishing, and supplies of water piped to homes and apartments, as well as the greater number of plumbing fixtures per dwelling and per building, have created an ever growing demand on the country's water resources. Indication of this rapid growth in demand is shown in Table I, which is quoted from Abrams.¹

Table I. Effect of Industrial Expansion on Water Consumption In Various Sections of U.S.A.

Location	Period	Estimated Increase of Population, Pct	Increase Water Consumption, Pct
Baltimore, Md.	1938 to 48	6.0	69*
Baton Rouge, La.	1937 to 45	6.0	78†
Detroit, Mich.	1932 to 45	4.3	47*
Galveston, Texas	1931 to 43	23.0	74†
Houston, Texas	1931 to 43	34.0	61†
Texas City, Texas	1931 to 43	63.5	2500†

* Consumption of municipal supply only.

† Consumption of both the municipal and private industrial supplies.

In commenting on the growing requirements of the State of Texas, the foregoing report states that "between 1890 and 1940 the population of the State

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Discussion on this paper, TP 3522F, may be sent (2 copies) to AIME before June 30, 1953. Manuscript, Nov. 23, 1952. Chicago Meeting, September 1952.

of Texas increased 287% and the water consumption 7000 per cent."

The domestic load varies greatly, depending on the extent of the available supply and the habits of the community. According to the report of the President's Water Resources Policy Commission² the United States Public Health Service reported a national average of 127 gal per capita per day, varying from 60 gal in communities of 500 persons to 140 or 150 gal in cities of 10,000 or over.

Other critical areas may be cited to illustrate the inter-relation between new industrial processes and the demands for water in the area where the new enterprise is located. Powell and Wilson³ have pointed out that during World War II, in the vicinity of Louisville, Ky., a rapid expansion of industry took place, consisting of numerous synthetic rubber plants. The demand for water increased from 37 million gal per day in 1937 to 62 million gal in 1943. These heavy requirements drew down the supply stored in the water-bearing aquifers to such an extent that urgent plans for recharging were considered. Similar situations have developed in the Los Angeles area and in the vicinity of Texas City.

Warne⁴ has drawn attention to the situation in Los Angeles area, where in the West Basin the withdrawal of ground water in 1945 was 90,000 acre-ft, almost double the amount of the natural fresh water recharge. The draw down has been so severe that the level of the ground water is now below sea level, and sea water is invading the West Basin aquifers at rates up to 300 ft per year.

In the Report of the Engineers Joint Council⁵ it is pointed out that in the area of Texas City the ground water table has been critically lowered. In 1930 the daily withdrawal from the water-bearing aquifer was 0.5 million gal. By 1945 the quantity withdrawn

had increased to 23 million gal per day. The level of the water table, therefore, had dropped approximately 150 ft below the sea level. As a result of draining the water-bearing sands, a subsidence of the land surface of 2.4 in. per year has taken place.

One of the most spectacular episodes of water loss brought about by man's activities is described as follows by Foose:⁶

In the vicinity of Hershey, and in the valley of Spring Creek there were at one time many springs, some of them yielding thousands of gallons per minute. One large spring in particular flowed forth to make several large ponds a few hundred feet east of the plant of the Hershey Chocolate Corporation. One of the reasons for the plant's location was this excellent source of water, known first as the Derry Spring. For more than a century the various springs were used on many of the farms in this area as their only water supply, and for nearly 50 years the Derry Spring has been used by the Hershey Chocolate Corporation. As far as it has been possible to determine, none of the large springs ever went dry, even during the longest droughts.

During the past fifty years many wells were drilled on farms and on properties where new homes were built. Except for very shallow wells, in which the water never stood more than several feet above the bottom of the well, none of these are known to have gone dry. A series of wells were also drilled by the expanding plant of the Hershey Chocolate Corporation. Yields ranged from 200 to 1200 gallons per minute, and a few had no yield at all, demonstrating how poor an aquifer limestone can be "in between the fractures." The Corporation wells continued to have a constant yield in wet and dry season alike, and they never affected in any way the adjacent springs or wells in the valley.

One and a quarter miles southwest of the Hershey Chocolate Corporation is a limestone quarrying operation about 125 feet deep. Relatively little ground water seeps into the quarry; for many years less than 1000 gallons per minute have had to be pumped to maintain operations. One and a half miles northeast of the Corporation is a larger limestone quarry and mining operation whose workings extend more than 400 feet below the surface. Until May, 1946, during normal operations about 3000 to 3500 gallons per minute of ground water were pumped out of the mine. Both of these mining operations remove limestone that is much purer than the average limestone of the valley. As a result there are many openings, large and small, that have been dissolved by water percolating through the rock fractures. Located on this same pure limestone formation is the Derry Spring. The trend of the limestone rocks in northeast-southwest is a nearly straight line connecting the spring and the large mining operation.

During routine mining operations in August, 1946, a blast in the hanging wall of the mine exposed a six-inch wide solution channel about 375 feet below the surface out of which poured an estimated 8,000 to 10,000 gallons of water per minute, flooding the mine in the course of a day. When this occurred nearby wells dried up, ground water seepage into an adjacent quarry ceased, the Derry Spring a mile and a half to the southwest dried up on the second day and two wells of the Hershey Chocolate Corporation were badly affected. After many months of labor the opening in the mine was sealed off with a large steel plate, the adjacent wells had water in them again, the flow of the Derry Spring was restored, and the Corporation wells were again normal.

Van Tuyl⁷ has described an interesting situation regarding the influence of air conditioning and modern

water uses on the water withdrawals from the aquifer underlying Pittsburgh. The strata are very permeable sands and gravels, deposited during the glacial age, and permit a rapid flow through the deposit. The storage capacity is, however, limited in extent, being approximately 800 million gal. The demand for cooling water has increased steadily since 1927, until at the present time it is double the quantity used in that year. The average withdrawal on a summer day has reached 10 million gal, which indicates a supply for only 80 days. Fortunately the recharge to the aquifer is continuous, although during the summer the draw down is from 8 to 10 ft. During winter, because of the proximity of the rivers and the ease of ground water flow, the water table is returned. The demand for water reaches a maximum in July and August, amounting to 25 pct of the total yearly volume of water pumped. During this period the demands for air condition equal the requirements of all other uses. In fact, the demands on the water resources of the aquifer have reached the maximum rate of recharge, and a further increase in water requirements cannot be satisfied from the present wells without artificial recharge. Natural recharge is limited by the prevention of rainfall soaking through the large percentage of the area of the triangle that is artificially impermeable by street paving, buildings, and parking lots.

Another notable man-made barrier is the sheet piling used to form the embankment along the Allegheny River. This wall, which is 3750 ft long and averages 33 ft deep, reduces the cross-sectional flow area into the aquifer to approximately 50 pct.

During the threatened water famine the metropolitan area of New York suffered an acute crisis, which taught the population in a most spectacular manner how dependent the individual and the community are on water in this present civilization. London, the site of which was selected by the ancient Britons because it was underlain with a porous gravel deposit providing a continuous source of water, has also in recent years been concerned with the lowering of the water table.

Table II. Typical Interstate Agreements for Control of Water Resources

COMPACT	STATES INVOLVED
The Interstate Commission on the Delaware River Basin (INCodel)	Pennsylvania; New York; New Jersey; Delaware
The Interstate Commission on the Potomac River Basin	Pennsylvania; Maryland; West Virginia; Virginia; District of Columbia; USA U.S. Public Health Service, Office of Army Engineers
The Ohio River Valley Water Sanitation Commission	New York; Pennsylvania; Virginia; West Virginia; Ohio; Kentucky; Indiana; Illinois

With these examples before the country, it is natural that all branches of government are taking a great interest in the conservation of water resources. The comprehensive report of the President's Water Resources Policy Commission⁸ is evidence of this concern at the Federal Government level.

There are many aspects of the problem, some in conflict with others: flood control, power generation, land reclamation, domestic and industrial water supply, water shed management, inland and tidal waterways, and stream pollution control. To balance these various factors will tax the combined skills of the country's outstanding engineers and economists.

Because the approach to the problem in many cases lies within the area contained in the drainage basin of the stream system, which never follows political boundary lines, interstate compacts have been adopted to facilitate the control of the water resources within the drainage area. Typical agreements are listed in Table II.

All commissions have been carrying out engineering and economic studies on which to base proposals for a well balanced project of water conservation and control in each of the drainage basins.

Recently, the proposed Incoel Interstate Water Project was made public.⁸ Briefly, it calls for the construction of a series of four reservoirs in the upper reaches of the Delaware River watershed. It will provide satisfactory sources of water for the areas that require it, and it will also provide an increase in the flow of the Delaware River during the dry months of summer and autumn. The salient features of the project are given in Table III.

Table III. Apportionment of Water Supply Under Proposed Incoel Interstate Water Project

Purpose	Million Gallons
Total reservoir storage capacity	527
Recreational use (boating, fishing, etc.)	111
Water supply to New York and northern New Jersey	100
Stream flow regulation and future water supply to Philadelphia and southern New Jersey	316

Because of the widespread demands, severe competition has developed between states, between communities, and between individuals, necessitating priorities for the use of water. Highest priority is assigned to water supplies for municipal and domestic purposes. All other factors must be modified to maintain the necessary standard of priority. It is this strict insistence on water purity for the protection of public health that has impelled industry to install equipment for the treatment of its water-borne trade wastes. Roughly, such wastes can be divided into two kinds, organic and inorganic. Typical of the former are the wastes arising from packing houses, food processing plants, pulp and paper mills, cheese and butter factories, and the beet sugar industry. The inorganic trade wastes include the pickle liquor from the steel industry, brines from various wells, acid water drainage from coal mines, and liquid wastes from the metal-finishing industry. Two other great sources of pollution are domestic sewage from centers of population, and solids eroded from the soil by action of annual rainfall run-off.

As mentioned previously, there is always some interaction between wastes discharged into the stream system. An interesting example given by Beal and Braley⁹ indicates the reluctance of communities to correct a situation until faced with a crisis:

There is a city not far from Pittsburgh that uses an open creek in lieu of a trunk sewer. The drainage from three large mines, ranging in volume according to season from four to seven million gallons daily, is also pumped into this creek. It is only the acidity of this mine drainage that inhibits putrefaction of the sewage. Now, with the possibility in the offing that mine pumping will be stopped, the city government is considering buying or leasing those abandoned workings in order that it may continue to add that acid water to the city's sewage, as a prophylactic, pending the time when the city government can complete a modern sewage disposal plant.

This commingling of sewage and mine drainage takes care of the area immediately surrounding this city just as other industrial wastes have their inhibiting effect on other cities' sewage. But as these streams flow on they receive other streams, often naturally alkaline, so that dilution and neutralization soon reduce the bactericidal and bacteriostatic effects of these industrial wastes. Because of this neutralization, new infection in the stream permits a resumption of sewage decomposition in areas that should be clean. Therefore, we may be sure that problems involved in the treatment of water-borne industrial wastes, rolling up like a snowball, are going to be among the tremendously important industrial problems of the next decade.

Biological Oxygen Demand

As pointed out by Eldridge¹⁰ the contamination of a stream by organic wastes arises from decomposition of those wastes through the action of bacteria. In consuming this organic material, the bacteria combine it with oxygen to maintain their life processes. This is *biological decomposition*. It is generally held that *anaerobic* bacteria carry out their work in the absence of oxygen. But as Eldridge has observed, these bacteria obtain their necessary oxygen from the organic compounds which are composed of such elements as oxygen, nitrogen, carbon, sulphur, and hydrogen. The removal of oxygen leaves elements that form compounds containing no oxygen, such as ammonia and hydrogen sulphide. These compounds have characteristic foul odors. When a body of water is dominated by such anaerobic decomposition, the aquatic life of the lake or stream is destroyed because of the toxicity of the compounds. If there is an ample supply of elemental oxygen present in the body of water, *aerobic* bacteria continue the work of decomposition to yield end products such as sulphates, nitrates, water, and carbon dioxide. The foul-smelling products created by the *anaerobic* decomposition are oxidized and are harmless. Thus the oxygen contained in the stream or lake is the most important factor in disposing of organic pollution discharged to the stream system. Where oxygen is not present in sufficient quantities to complete the decomposition of organic wastes, the stream waters may be completely depleted of oxygen, whereupon the stream will become septic.

This demand for oxygen for the decomposition of organic by biochemical processes has resulted in a measure of pollution universally employed by sanitary engineers. It is known as the Biochemical Oxygen Demand, or abbreviated, B.O.D. It is defined by Phelps¹¹ as "the oxygen that will be demanded by the material in the course of its complete oxidation biochemically. It is not at all related to the complete oxygen requirements in chemical combustion, but is determined wholly by the availability of the material as a bacterial food and by the amount of oxygen utilized by the bacteria during its oxidation." This measure, while of great importance to such enterprises as the food processing industries, pulp and paper mills, tanneries, creameries, and domestic sewage disposal works, does not apply to such inorganic wastes as acid water drainage from coal mines and suspended solids contained in the water discharged from coal preparation plants. However, it is of interest to note that in one set of standards the measure of B.O.D. was to be applied to all wastes. That requirement has been changed.

Stream pollution created by the coal industry

stems from two main sources, acid water drainage from mines and suspended solids discharged with the effluent from coal preparation plants.

Acid Mine Drainage

The acid mine water problem has been subjected to intensive research for years, and the investigations are continuing at Mellon Institute, West Virginia University, and Johns Hopkins University. It is surrounded with great difficulties, not the least of which are the stupendous volumes of water that must be handled.

According to Supplement C of the Ohio River Pollution Survey,¹² the total estimated mine acid load delivered to the Ohio River Drainage Basin was approximately 2,500,000 tons per year, a figure derived from the data collected in 1940. Since that time numerous mines have been worked out and abandoned and new operations started. The measure of confidence that can be placed in the figure, therefore, is not known, but it will serve to indicate the magnitude of the problem.

In the Mellon Institute *Survey of Mine Drainage*,¹³ Dr. S. A. Braley points out:

In general, acid mine drainage consists of a solution of iron, aluminum, calcium and magnesium sulphates. The iron and aluminum sulphates react with water to give an acid solution, whereas the calcium and magnesium sulphates contribute to the hardness. The acid coming from the iron and aluminum sulphates, neutralizes the natural alkalinity of the stream receiving the drainage.

The iron in the drainage may cause the receiving stream to become very turbid with iron hydroxide or, depending on the concentration of iron sulphates in the drainage, may result in a red water where the iron remains dissolved in the highly acid solution.

The mine drainage in western Pennsylvania, West Virginia, Ohio, and Kentucky, where many of the mines are shallow, is greatly affected by rainfall. Results shown in Table IV are taken from a study

Table IV. Effect of Rainfall on Mine Drainage in Monongahela and Cheat River Area

Year	Gal Per Day Per Acre of Mined Out Area	Remarks
1928 to 1929	1000	Normal
1929 to 1930	500	Drought latter 1929, entire year 1930
1931	660	Normal

made by Carpenter and Herndon of mines located on the water sheds of the Monongahela and Cheat Rivers.¹⁴ The average cover over most of these mines varied from 100 to 300 ft. Braley¹⁵ in his work on mine acid drainage at Mellon Institute has noted a similar occurrence. In mines having shallow cover, the changes in the rate of drainage flow are closely parallel to the flow of the surface streams of the neighborhood, but the flow variation becomes more uniform as the depth of cover increases. Unfortunately, the greatest difficulty is experienced in the mines that have shallow cover, especially if their discharge is above the stream level, as the drainage ultimately reaches the water shed.

In abandoned shallow cover drift mines, the fluctuation of flow can vary as greatly as 25 to 1 in a few days and a ratio of 6 to 1 in volume of water can take place in a few hours. For example, follow-

ing a heavy rainfall the flow was observed to rise from 50,000 to 700,000 gal per day in 4 hr, a ratio of 12.7 to 1. To design any kind of treatment that would automatically adjust itself to such a wide range would tax engineering ability to the utmost. Economically it would not be feasible, and in the case of abandoned mines it is out of the question. Yet the greatest volume of acid water discharge is from such mines.

A comment may be made here regarding the use of the pH measure in the study of acid mine drainage. Carpenter and Herndon¹⁴ and Braley¹⁵ all noted that in some cases the pH value of the stream does not rise after dilution with the alkaline waters from tributary streams. That is true for most of the acid mine discharges in western Pennsylvania and northern West Virginia. A buffering action seems to take place at an approximate pH value of 3.0 that permits appreciable quantities of alkali to be added to the acid solutions, without raising the pH. Selections from Table III in Carpenter and Herndon,¹⁴ shown here in Table V, indicate this action. The

Table V. Relation Between Dilution and pH Value

Sample No.	6	7	8	11	12
Initial Acidity in P.P.M.	1200	3600	920	950	7100
Ratio of Dilution	pH Values				
0	3.0	3.0	3.0	3.0	3.0
1 to 1	3.0	3.0	3.0	3.0	3.0
1 to 2	3.1	3.0	3.0	3.0	3.0
1 to 5	3.2	3.0	3.2	3.3	3.0
1 to 10		3.0	3.2	3.3	3.0
1 to 20		3.0	3.8		
1 to 30					
1 to 40					
1 to 50	3.6	3.5	4.2	4.2	3.0
1 to 100	4.0	3.5	4.9	5.0	3.5
1 to 200	4.4	3.8	5.5		
1 to 400					

samples were taken in northern West Virginia. Because of the existence of buffering action those investigating acid mine water problems are aware that the pH value may lead to false conclusions. That is not to say, however, that the pH value is not useful. It is generally determined and reported. It represents the hydrogen ion concentration at an equilibrium depending upon concentration and temperature, and is not a measure of the titratable acidity. The acidity of acid mine drainage is the result of the hydrolysis of the iron and aluminum salts into free acid and hydroxides and/or basic sulphates, and the total titratable acidity is the result of their complete hydrolysis.

During their studies of the forms in which sulphur occurs in coal, Parr and Powell¹⁶ indicated that bacterial action by some of the sulphur-producing organisms might possibly explain some of the high acidities found in mine waste waters. Investigation of this possibility was carried on by Hinkle and Koehler¹⁷ at West Virginia University through the employment of bacteriological techniques. Two micro-organisms have been isolated that are thought to have a part in promoting the formation of acid. One of these, *Thiobacillus Thiooxidans*, converts elemental sulphur or sodium thiosulphate to sulphuric acid. Further study is being carried on to determine the reasons for the rapid formation of acid in abandoned mines compared to those in operation. In so far as the investigations have been carried on, these micro-organisms have not been found in alkaline mine waters.

The other organism seems to be a factor in the oxidation of ferrous sulphate to ferric sulphate with the subsequent hydrolysis to ferric hydroxide.

Temple and Colmer,¹⁸ in continuing the work at West Virginia University, showed that the oxidation of ferrous iron in acid water to produce the characteristic red color was a bacteriological process and not one of simple atmospheric oxidation. They suggested that the bacterium be designated as *Thiobacillus Ferrooxidans*.

This phase of research is still subject to discussion and consideration by the various investigators as the work proceeds. Leathen and Braley⁹ have investigated this aspect of the problem independently and have commented as follows:

Both sulfur and iron oxidizing bacteria have been isolated from all of the acid mine waters examined. The sulfur oxidizing bacterium has been identified as *Thiobacillus thiooxidans*, and oxidizes elemental sulfur to sulfuric acid. One statement has been found in the literature to the effect that *Thiobacillus thiooxidans* may oxidize sulfide sulfur, but that this has not been confirmed experimentally. It is believed that Mellon Institute may properly be credited with providing some experimental evidence that this organism does not oxidize sulfide sulfur, at least as it occurs in the coal measures. For, in the experiments, when graded sulfur-ball material replaced elemental sulfur in the substrates, and the substrates were inoculated with the microorganism, the normal, chemical rate of acid formation was not enhanced. Inoculation of media, containing museum grade pyrite, also, was not oxidized by these bacteria. Inoculated substrates containing museum grade marcasite, however, indicated a slight increase in acid formation which could be attributed to *Thiobacillus thiooxidans*. It is not felt that appreciable formation of acid in a mine can be attributed to this microorganism, as the sulfuritic material, "sulfur-ball", usually found in bituminous coal seams and associated rock strata are not oxidized by any of the strains of *Thiobacillus thiooxidans* used in our studies.

The iron oxidizing bacteria, which this Institute is refraining from classifying until the physiological studies are completed, have caused three to five fold increases in the amount of acid formed from sulfur-ball material and from marcasite. Museum grade pyrite was not attacked. At the present time, the amount of acid produced from sulfuritic materials by this microorganism in nature is unknown, and cannot be differentiated from that produced by strictly chemical reactions.

Recently, Dr. Jay V. Beck, Brigham Young University, has written that he has been able to demonstrate that our cultures, as well as some he obtained from Bingham Canyon, Utah, were "... able to convert iron pyrite to soluble iron with a decrease in pH." This observation confirms some of our work.

It is suggested that the role of the iron oxidizing bacteria in acid production may be as follows: the speed of atmospheric oxidation of the iron sulfides seems to vary directly as the available surface, the amorphous sulfur ball oxidizing with much greater rapidity than the densely crystalline pyrite. Ferrous sulfate, the product of the first atmospheric oxidation, is in chemical equilibrium with the sulfides. The iron oxidizing bacteria then oxidize ferrous sulfate to ferric sulfate, which, in contact with sulfur ball material, oxidizes the latter to ferrous sulfate while the ferric sulfate is in turn reduced to ferrous sulfate. This increased quantity of ferrous sulfate now undergoes bacterial oxidation, and the cycle repeats and repeats in the manner of an expanding spiral.

The iron oxidizing bacteria, however, should not be considered wholly detrimental in nature, but are of assistance in the deposition of ferric sulfate in streams, outside of the mine. It has been determined that concentrations of ferrous iron, oxidized

only 48 percent by a strictly chemical reaction in two years, were completely oxidized by these bacteria in eight days. Such oxidations are advantageous.

The iron oxidizing bacteria, by greatly increasing the rate of oxidation, induces the deposition of basic ferric sulfate, 'yellow boy', in the shortest possible distance in the stream, confining an unsightly condition to the smallest possible area. Furthermore, it greatly assists the stream to make a very rapid recovery as far as dissolved oxygen is concerned. Dissolved oxygen is essential to all aquatic life, both plant and animal. When acid, ferrous-iron-bearing, waters enter a normal stream, the dissolved oxygen is practically depleted owing to the amount required to transfer the ferrous iron to the ferric state. If this were to take place at the slow chemical rate of oxidation, whole streams, or at least huge portions of them, would be void of all life. When the oxidation rate is increased several fold by the activity of the bacteria, this area, depleted of oxygen and void of aquatic life, is confined to the smallest possible section of the stream. The stream, then, has an opportunity to absorb atmospheric oxygen to replenish its supply. Making a rapid recovery, the stream can again support an abundance of both plant and animal life in a short distance from where the ferruginous water entered. The organism, in this respect, is of distinct value in nature.

It is frequently stated by many who have only given cursory thought and consideration that the problem can readily be solved by neutralizing with lime. In all probability, this method of attacking such a situation stems from the installation at the Calumet Mine of the H. C. Frick Coal Co. It was installed in 1914 during World War I to produce certain specific products for gas purification that were unavailable from Germany at the time. A description of the plant is given by L. D. Tracy in the Transactions of the AIME in 1921.¹⁹ It was abandoned immediately after the War.

To ascertain the feasibility of lime treating acid mine water, the Pennsylvania Sanitary Water Board authorized the Mellon Institute to carry out research on lime in its various forms. The following conclusions were based on the aforementioned research. They are published in a report issued by the Pennsylvania Sanitary Water Board, Department of Health, Commonwealth of Pennsylvania:²⁰

1. Although acid mine drainage can be chemically treated with lime or other alkalies to neutralize the acid, such a method is not practical or feasible because of the economic and other difficulties involved.
2. The use of limestone or hydrated lime to neutralize acid mine drainage produces hard water.
3. A neutralizing treatment would be effective only if the plant were designed and staffed to treat the maximum seasonal flows and the large fluctuations caused by natural conditions.
4. After completion of mining operations on a property, continuation of the treatment indefinitely would be required.
5. The findings of the Mellon Institute Fellowship program to date show that the water that enters a mine through rock and earth strata is normal ground water, free from acid. From these results, it seems quite evident that the only hope of preventing the flow of acid drainage from a mine is by:
 - a. Stopping the formation of acid from sulfuritic material on the walls and roof, or—
 - b. causing the water entering the mine to

leave the mine by the shortest possible route and quickest possible time without leaching the acidic substances so formed from the walls and roof, or—

- c. diverting all water from the mine. The Fellowship referred to has no confidence in any proposal to neutralize continuously the acid mine water flowing from a coal mine in Western Pennsylvania.
6. Notwithstanding the impracticability of the suggested limestone or lime treatments for continuous flow of acid drainage from mines, hydrated lime might be used successfully to neutralize casual pools of acid water found in coal workings.

In regard to the practicability of using limestone or hydrated lime, theoretically, it would require approximately 1 ton of lime to neutralize 1 ton of sulphuric acid in acid mine water. It would require a large excess of lime to complete the reaction in a reasonable time.

In regard to the availability of lime to treat these acid waters, statistics from the Minerals Year Book, 1944,²¹ may be of interest, see Table VI. According to Supplement C of the Ohio River Pollution Survey¹² the acid load discharged into the Ohio River streams system from Pennsylvania approximates 889,000 tons per year. Possibly it would take all the lime that Pennsylvania produces to treat the situation. Even should there be sufficient lime to treat the acid drainage, the oxides of iron would separate as a watery gelatinous sludge which settles very slowly. If this material were to be settled before the treated drainage was discharged to the stream system, very extensive lagooning acreage would be required. In hilly terrain, similar to western Pennsylvania, West Virginia, and East Kentucky, such lagooning would present great difficulty. From time to time, such settling basins would have to be cleaned out. That poses the extremely difficult problem of disposal.

Table VI. Availability of Lime

Year	Market	Tons
1944	U.S.A. production open market and captive	6,473,563
1944	U.S.A. Total open market-captive-chemical plants	8,954,183
1944	Penna. Production Agric.-Bldg.-Ind.-Chem.-Refr.	1,026,292

Some study was made to ascertain if a market could be found in the paint industry for the ferric oxide, or rouge, but the results were not promising. On the other hand, if the material were deposited on the adjacent hills, the 36-in. annual rainfall would wash the sludge back into the stream system. There is another result of such a lime treatment. In the reactions that neutralize the acid, the iron in the acid mine water is replaced by calcium and all the sulphate originally present remains in solution as calcium sulphate. This material imparts permanent hardness to the water.

In the anthracite field considerable trouble is experienced with acid drainage water, and in many cases where a percentage of the volume pumped is diverted for use in the coal preparation plant or breaker the water is treated with lime for neutralization before use in the breaker equipment.

The problem of corrosion of equipment is in some cases very severe. As a result many of the companies have resorted to the use of acid-resistant metals and extra heavy dimensions in the design of the pumps, valves, and piping.

According to Ash,²² the treatment of acid mine water for the coal cleaning plant through the use of lime has been in practice since 1932. A concentrated lime water mixture or slurry is fed either into the breaker water supply reservoir or into the pumping system. High calcium lime + 90 pct CaO, magnesium lime, MgO 5 to 25 pct, dolomitic limes, MgO 25 to 45 pct, and hydrated lime, Ca(OH)₂, are used. The rates of reaction of high calcium hydrated lime and of dolomitic lime are rapid. Sulphuric acid, however, forms insoluble calcium sulphate and this retards the reaction. With the dolomite magnesium sulphate is formed and is more soluble in water.

It must always be borne in mind that in the anthracite field of Pennsylvania, the amount of water that must be removed from the operations is extremely great, averaging 30 to 40 tons per ton of coal produced. As Griffith²³ has pointed out, although the average rainfall in the region has not altered, the amount of water entering the mines through breakage of the overlying strata to the surface is continually increasing. An example is

Table VII. Water Removal in Typical Anthracite Mining Operation

Years	Ratio, Tons of Water Pumped to Tons of Coal Produced
1920	8.4 to 1
1925	10.6 to 1
1930	11.4 to 1
1935	26.2 to 1
1940	32.7 to 1
1942	30.3 to 1

given in Table VII of one of the larger operations in the northern anthracite field.

The difficult aspect of the problem is to make provision for controlling the drainage of abandoned or inoperative mines. Of the total volume pumped, 66 pct is discharged from the aforementioned class of properties. Under such conditions, with the economic factors involved, comprehensive planning is needed for the handling of the water of the whole anthracite region. Investigations are being carried out. One method seems the most feasible, i.e., a tunnel system arranged for draining by gravity those reservoirs that lie above the tunnel outlet and lifting mechanically the water lying below the outlet. The plans must be long range, however, as the number of inoperative and abandoned mines are on the increase. However, it should be remembered that the volume of drainage from the anthracite region, 730 sec ft, is great enough to affect the volume of flow in the surface stream system, if it is removed. It is a phase that must be taken into account.

The Suspended Solids in Washery Waters

Although at the present time "no practicable method of removing the acid properties of mine drainage is known," as Felegy, Johnson, and Westfield point out,²⁴ there are some methods that can be employed for the treatment of suspended solids from water discharged to the stream system from coal preparation plants. The problem of suspended solids is the second major difficulty facing the coal industry in the control of stream pollution.

The silt problem created by the anthracite industry from water discharged from the breakers has been one of long standing, and it is only in recent years, with the cooperation of the Department of Engineering of the Pennsylvania Department of

Health, that the removal of suspended solids has progressed.

The water clarification systems of various coal preparation plants have been described in detail in numerous articles, and as each one has been designed to meet the individual needs of the particular plant, no attempt will be made in this paper to cover them. Nearly all the systems employ some or all of the following equipment.

In elevated settling cones the suspended solids settle toward the apex and are drawn off as a thickened slurry or sludge, while the clarified water overflows at the surface into a launder around the periphery to a discharge funnel. There are no moving parts. Gravity and the hydrostatic head are depended on to maintain the flow of the thickened sludge.

Drag conveyor settling tanks are equipped with a slow moving scraper conveyor that drags along the bottom of the tank, then up an incline to a lip at the end of the tank. The settled material is thus removed from the tank, while the clarified water is discharged into a sump.

Dorr thickeners are generally circular in shape with a conical bottom sloping gently toward the center. A series of ploughs attached to a rotating supporting structure moves the settled sludge spiral fashion toward the center of the apparatus. The sludge is then withdrawn in its thickened state through special pumps. The clarified water overflows the launder constructed around the circumference of the unit, which is quite flexible.

Dependent on the rate of operation, this equipment may be used as a classifier removing the coarser particles, as an underflow, and overflowing the finer particles with the water into the launder.

Hydraulic cyclones developed by the Dutch coal industry in Holland have recently been applied to the clarification of washery water. This equipment, which also acts as a fine coal cleaning unit, has many applications.

For the further dewatering of the thickened sludge, centrifuges, vacuum filters, and thermal dryers are employed. The necessary pumps and piping, of course, form part of the system.

Finally, at many coal preparation plants, some or all of the water overflow from the clarification system is discharged to settling ponds or lagoons for a long period of retention prior to being discharged to the stream system. In some instances such settling basins may be cleaned out periodically, while in others the acreage and storage capacity are great enough to serve for years.

It is safe to say that with the growing demand for a reduction in the quantity of suspended solids in the washery effluent, greater care and thought must be given to the design, and more elaborate equipment will have to be installed. In passing, it may be said that the designer is not entirely responsible for the inability of the clarification system to remove sufficient solids. If the plant is designed for a stipulated capacity, and the owner increases the feed appreciably, it is virtually certain that the settling system will be overloaded. Another factor is the cost of the plant. The initial design may have provided ample equipment to yield a satisfactory effluent. However, when the estimate of cost is received, there is an immediate request for a revision of the estimate downwards. Such revisions result in the removal of some equipment from the design.

In the future, greater attention will have to be

paid both by the designer of plants and the coal operator to the requirements of the regulatory bodies having control of stream pollution. At the present time there is under consideration a tentative suggestion from the Pennsylvania Department of Public Health regarding the amount of suspended solids discharged to the stream system from coal cleaning plants. It is realized that to require complete removal of all suspended matter from the water is impractical, so it was suggested that an allowance of 8 lb of —325 mesh suspended solids per ton of clean coal produced be permitted in the washery effluent discharged to the stream system. A number of competent engineers experienced in the design and operation of coal preparation plants have studied and commented on the measure. That it could present difficulties is illustrated in the criticism by Parmley,²⁵ who writes that the following practical difficulties should be given consideration:

Case 1.

A mine with a R.O.M. feed of 400 T.P.H. with 35% refuse containing considerable fire clay and material that will disintegrate in water. Compared with a mine with 400 T.P.H. 10% reject and a solid shale reject. The one with 35% reject will have 260 tons of clean coal and is allowed only 2080 lbs. per hour waste. In this case the operator would have a very difficult clarification problem, yet is penalized and allowed less waste material than the operator with 10% reject. In the case of 400 T.P.H. with 10% reject there will be produced 360 tons of clean coal with an allowance of 2880 lbs. per hour. In all likelihood with this case the clarification system would be very efficient and there would be less than 8 lbs. per ton wasted.

Case 2.

Take two operators with 400 T.P.H. R.O.M.—one with 40% minus $\frac{1}{4}$ " material and the other with 20% minus $\frac{1}{4}$ ". The one with 40% minus $\frac{1}{4}$ " would have a very difficult settling problem yet would be allowed only the same wastage as the one with 20% minus $\frac{1}{4}$ ".

Case 3.

The most glaring example is a plant with a capacity of approximately 2400 T.P.H. feed and 30% reject, giving a clean coal of 1680 T.P.H. There would be a wastage of 11840# (5.92 tons) per hour or 83 tons per 14 hour day. This is a lot of material to run to the stream. The question is how can this tonnage be wasted to the river. If wasted with water at the rate of 200 G.P.M. with 11840# per hour or 197 lbs. per minute of solids would give a water containing 10.6% solids. This wastage would be very black and no state inspector would allow such a wastage to go to the river. Yet, according to the tonnage standard, it would be legal.

Now, consider wasting in say 4000 gal. per min. (which would be very impractical), the % solids in the wastage would be $\frac{1}{2}$ of 1%. With only $\frac{1}{2}$ of 1% solids, the waste water would be gray and would probably be allowed to flow into the stream.

Case 4.

Take an operator with 200 T.P.H. and 15% reject or 170 T.P.H. of clean coal. The allowable wastage would be 1360 lbs. per hour or 23 lbs. per minute.

The amount of water wasted would probably depend on the operator's water source or scarcity. Wasted with 23 G.P.M. of water would give 10% solids. Due to blackness of the wasted water, it would not be allowed. If the operator had to waste the same amount of solids upon a visually approved wastage of $\frac{1}{2}$ of 1%, it would be necessary to waste at the rate of 500 G.P.M. This would likely be

prohibitive and uneconomical from the operator's standpoint.

It is realized that in all of the above cases the state would say that although the standards would be met, it would be up to the operator to install clarification equipment or waste the higher gallonage. It would be up to him to choose his method.

It is felt that a standard should be based on the % solids in the wasted water and limited to a maximum top size in the solids.

Just what criteria will be used it is hard to say, but that one will be adopted is certain.

The problem of a sliming clay, mentioned in Case 1, has an important bearing on the method of clarification. During recent years, the trend in the bituminous coal fields has been toward full seam mining, employing completely mechanized methods. A decade or more ago the quantity of refuse discharged from the coal preparation plant was approximately 15 to 20 pct of the raw feed. The modern plant must dispose of 25 to 35 pct of the raw feed as refuse, which in many cases contains clay material that breaks down in the wash water into fine sliming materials approaching a colloidal condition. When such situations are encountered, the washery water becomes heavily overloaded with suspended solids that are very difficult to settle in any reasonable retention time. However, that rate of settling has been increased through the use of flocculants that cause the clay to form flocs. The treatment has been varied to suit local conditions, but it is adequately covered in the literature by Samuel²⁶ and many others.

The acid water drainage and the discharge of washery water carrying suspended solids are the major problems encountered in the coal industry. There are other minor difficulties such as the oil slick carried off by rainfall where dust-proofing oil employed at the loading booms of the preparation plant has spilled on the ground. These are local occurrences, however, and can easily be corrected.

It is quite apparent that the coal industry is becoming aware of the demand for stream pollution control and is contributing to the research programs that are being carried on at various institutions in an effort to find some solution to acid water. To date no satisfactory treatment has been found.

Legal Controls on Stream Pollution

Industry and municipalities are becoming better informed on the obligations placed upon them through the recent legislation governing water supply and stream pollution. They are in more frequent contact with the officials of the commissions and boards that have been appointed to administer the various State, Interstate, and Federal laws and compacts. At the Federal level, Public Law 845, the 80th Congress, was adopted "to provide for water pollution control activities in the Public Health Service of the Federal Security Agency and for other purposes." Since this law was passed, the Public Health Service Agency has become increasingly active in all phases of stream pollution, and its activities will continue.

The Ohio River Valley Water Sanitation Compact, ratified by the legislatures of the eight states embraced by the Ohio River basin, outlines in the Preamble the reasons for the compact, namely:

The rapid increase in the population of the various metropolitan areas situated within the Ohio drainage basin, and the growth in industrial activity within that area, have resulted in recent years in an increasingly serious pollution of the waters

and streams within the said drainage basin, constituting a grave menace to the health, welfare and recreational facilities of the people living in such basin, and occasioning great economic loss.

The control of future pollution and the abatement of existing pollution in the waters of said basin are of prime importance to the people thereof, and can best be accomplished through the cooperation of the States situated therein, by and through a joint or common agency.

The Interstate agreement between the various states in the Potomac River basin is similar in intent to the foregoing quotation.

The Commonwealth of Pennsylvania adopted a Clean Streams Act of 1937. It was amended in 1945 to include culm and silt, or suspended solids, from coal mines and coal cleaning plants. In this law, *pollution* is construed to mean the discharge to or the effects on the stream system of noxious and deleterious substances, "rendering unclean the waters of the Commonwealth to the extent of being harmful or inimical to the public health, or to animal or aquatic life, or to the use of such waters for domestic water supply, or industrial purposes, or for recreation."

Industrial wastes are construed to mean "any liquid, gaseous or solid substance, not sewage, resulting from any manufacturing or industry, or from any establishment, which causes pollution as defined, and silt, coal mine solids, rock, debris, dirt and clay from coal mines, coal collieries, breakers, or other coal processing operations."

At least 45 states have adopted some type of legislation for the conservation of water resources and the control of stream pollution. It is interesting to note that many of the states have patterned their laws on many of the clauses contained in the Pennsylvania Act No. 177, 1945, the Brunner Bill.

The enforcement of these stream pollution control laws is generally assigned to regulatory bodies or commissions appointed by the governors of the individual states. Generally the regulatory bodies are vested with almost unlimited powers. On the other hand, it is unfortunate that there is no system of checks and limitations on their acts and requirements and that they cannot be held responsible for any economic effects which may be brought about by the fulfillment of their requirements.

Wachter²⁷ has drawn attention to this potentially dangerous situation:

It is characteristic of pollution abatement laws that the Commissions are not required to relate abatement requirements to actual or potential human uses of the streams. They customarily fix standards which relate theoretical stream conditions to theoretical possible uses, but are not required to relate specific requirements to actual conditions and uses. Consequently, many of them, under the laws by which they operate, have the power to require unrealistic and unnecessarily expensive degrees of treatment whether human purposes are served or not.

In such circumstances the success of the whole control program depends on the sagacity of the members of the commission.

Industry must keep fully informed of the various actions and decisions handed down by the Courts in relation to stream pollution. A case of far-reaching importance in its implications was decided by the Pennsylvania State Supreme Court. A coal mining company, using stripping methods, commenced operations on a clean stream water shed. The State Sanitary Water Board sought an injunction against

the company in Dauphin County Court, since mining operations were started without a permit. The Court ruled that mining could continue, if the streams were not polluted, and ignored the contention that a permit was necessary. The Supreme Court reversed the decision on the permit question, and ruled that the Commonwealth was justified in seeking assurance that any new mining operations will not destroy the purity of a stream before issuing a permit. Thus it is practically incumbent on the Sanitary Water Board to assure itself that no new coal stripping operation is a potential source of pollution on a clean stream.

Such a responsibility, "if the Law allows it and the Court awards it," can create inequities that may cause great hurt. Two examples may be cited.

An owner of land acreage invested in rural acreage and purchased with it the mineral rights of any underlying coal seam. Subsequently, on his applying for a permit to mine the coal, the permit was denied because of a potential pollution hazard. Therefore the owner could not realize, or retrieve, his investment.

The second case is a marginal mine that is a source of employment for an isolated community of 900 persons. The property is operated at a loss, but is kept going because of the community. The mine-washing plant is polluting a nearby stream with suspended solids. Any capital expenditure to remove the solids from the water discharged cannot be justified. Economics would require that the whole operation be abandoned, and the livelihood of the community would be at an end.

Such questions as these would tax the wisdom of the members of any commission. The economic effects of all abatement programs should be carefully weighed, and the relative benefits of each scheme should be compared to assure the community the most balanced program for all its needs. Such decisions call for rare judgment. Determining the relative importance of the esthetic and recreational values as compared to local economic considerations is especially difficult but frequently necessary. Employing rational means for attacking the problem may leave a lot to be desired because the influence of the emotional and psychological bias of society plays a great part in the decision.

Little has been said regarding the equally formidable problem of providing adequate sewage treatment plants for centers of population. Many of the great cities of the USA have no treatment plants at all. The human wastes from numerous communities are discharged directly into the stream system or to tidewater. To install modern plants for centers like Pittsburgh, Louisville, or Cincinnati is a stupendous task and requires sound financial planning. Engineering techniques are, however, well known.

Erosion caused by run-off of rainfall has not been touched upon, although the stream systems of the country carry stupendous quantities of silt to the sea. Radioactive substances in the sewer systems that receive the wastes from hospitals in city centers are also a factor. Time does not permit consideration of these and many other conditions.

Conclusion

The method of attacking the whole problem of stream pollution is made extremely difficult because of the meager knowledge of the specific effects of the numerous wastes of various quantities and concentrations that enter the river basin. Little is

known of their action on aquatic life, nor the interaction between themselves. A great field of investigation lies ahead, and it is to be hoped that greater knowledge will be available in the coming years for guidance of the planners and the administrators of these comprehensive programs for the conservation of the country's water resources.

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Upgrading Domestic Manganese Ores By Leaching with Caustic Soda

by R. V. Lundquist

Leaching manganese-bearing materials with NaOH to remove caustic-soluble silica has been demonstrated as a method for upgrading manganese. Those materials containing opaline varieties of silica respond most readily. Spent leach liquors are regenerated by treatment with lime to precipitate calcium silicate and to re-activate the NaOH for leaching.

ALL of our larger domestic deposits of manganese are very low grade, but estimated tonnages in these deposits are large enough to constitute a potential strategic reserve of manganese for emergency periods, provided these ores can be upgraded sufficiently to meet industrial specifications. Many types of processes have been suggested for upgrading these ores, several have been studied in pilot plants, and a few have attained limited commercial production for special purposes.

The hydrometallurgical process to be described below proposes a leaching operation for removing caustic-soluble silica and other gangue minerals and consequently upgrading a manganese ore or flotation concentrate to a commercially acceptable product. It is known that some forms of silica and some silicate minerals are soluble in NaOH solutions under suitable conditions. This new process proposes to use NaOH for removing the soluble materials from manganese ores or concentrates and, as a result, upgrade the manganese content. Since spent leach liquors are not efficient leaching agents the liquor must be treated with lime to precipitate calcium silicate and reactivate the NaOH.

Experimental Procedure

Manganese-ore samples ground to -35 mesh were leached in iron containers into which the ore sample and appropriate NaOH solutions were placed. Flotation concentrates were already ground to -200 mesh. The slurries were then placed on a hot plate or into a pressure digester and stirred continuously during the period of digestion. Where the digestion was performed at atmospheric pressures, water lost by evaporation was replaced at regular intervals to maintain approximately a constant volume of slurry. In the pressure digester, volume changes were not

controlled. After digestion, the slurry was allowed to cool enough for easy handling, but it was filtered hot, 50° to 70°C, on a Buechner filter. Filtration under these conditions was satisfactory. The resulting filter cake was washed and dried.

In most instances, a single leach would not produce maximum extraction of silica from the ore sample; consequently, the dried tailing was treated again in the same manner with a fresh solution of NaOH to obtain an additional extraction of silica. This procedure was repeated a number of times to get the maximum extraction of silica. Usually three or at most four stages of leaching were adequate for this purpose.

Table I. Analysis of Artillery Peak Flotation Concentrate

Item	Pct
Mn	31.65
SiO ₂	20.97
Al ₂ O ₃	3.12
FeO	3.11
CaO	3.24
BaO	8.02
K ₂ O	1.73
Pb	0.93
SO ₄	0.66

This leaching procedure was used for obtaining the data reported in this paper on the effects of NaOH concentration, pulp dilution, temperature, and time of digestion on the efficiency of extraction of silica from the ore sample.

The experimental procedure does not match industrial methods. Consequently tests for counter-current leaching were set up. Data from these tests are not yet complete, but in a qualitative way they indicate that thickeners between stages of leaching will be satisfactory and that the quantity of silica in a leach liquor moving forward to a new solid has only a minor effect on the quantity of silica extracted from the new solid. A pilot plant is now being designed for operation in the near future to test and evaluate the merits of this process.

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Discussion on this paper, TP 3486B, may be sent (2 copies) to AIME before June 30, 1953. Manuscript, Sept. 26, 1951. Revised, Nov. 21, 1952.

Table II. Effects of Time and Temperature on Leaching Artillery Peak Flotation Concentrates, First-Stage Leach

Test Time, Hr	Temp, °C	Initial Pulp Dilution	NaOH, G Per Liter	Extraction, Pct		Analysis of Tailings, Pct			
				SiO ₂	Al ₂ O ₃	Mn	SiO ₂	Al ₂ O ₃	
105A	1	160	2 to 1	401.9	61.2	6.3	36.27	9.40	3.38
105B	1 1/3	160	2 to 1	401.9	64.6	9.0	36.07	8.64	3.30
105C	2	160	2 to 1	401.9	65.4	7.2	36.37	8.40	3.36
105D	2 1/2	160	2 to 1	401.9	63.8	9.0	36.67	8.84	3.30
87	4 1/4	160	2 to 1	383.2	67.0	12.4	36.30	8.12	3.20
83	1 1/6	160	2 to 1	412.8	64.1	5.5	36.20	8.68	3.40
74A	1/2	90	2 to 1	397.2	39.8	5.3	34.63	14.14	3.31
74B	1	90	2 to 1	397.2	42.3	0	35.00	13.44	3.61
74C	1 1/2	90	2 to 1	397.2	45.1	0	35.10	12.86	3.55
74D	2	90	2 to 1	397.2	48.3	6.1	35.50	12.12	3.28
70A	4	91	2 to 1	398.4	47.7		36.00	12.30	

Artillery Peak Experiment. Kingman, Mojave Co., Ariz.: A flotation concentrate from this ore was used to determine the optimum conditions for leaching with NaOH solutions. This flotation concentrate contained quartz, chalcedony and opal as its major gangue constituents, feldspar and calcite as intermediate constituents, and amphibole, limonite, barite, clay, and gypsum as minor constituents. Analysis is shown in Table I. This concentrate was ignited at 1200°C and the manganese content thus increased to 36.57 pct.

Table II shows the effect of time and temperature on leaching. At 90°C, maximum extraction for a single leach was obtained after 2 hr of leaching when 48.3 pct of the silica was extracted. Under similar conditions, but at a temperature of 160°C, which corresponds to a steam pressure of 75 psi, 65.4 pct of the silica was extracted. Longer periods of leaching did not materially increase the extraction, but increased temperature markedly improved the extraction of silica.

In this paper, pulp dilution or ratio of solution to ore is expressed as the volume of liquor in milli-

liters per unit weight of concentrates in grams. This avoids the necessity of considering the specific gravity of the solutions each time. When 30 pct NaOH, 398.4 g per liter, solution is used the corresponding approximate weight ratios and percent solids are as follows:

Pulp Dilution	Wt Liquid Wt Solids	Solids in Pulp, Pct
2 to 1	2.65	27
3 to 1	3.98	20
6 to 1	7.95	11

Table III shows the effect of pulp dilution and the number of stages of leaching on the extraction of silica. Here it is noted that the more dilute pulps showed the best extractions of silica, also, that each additional stage of leaching increased the extraction, and the increased extraction per stage decreased as the number of stages increased until maximum extraction had been obtained. Further, a single stage of leaching at 160°C produced approximately the same extraction of silica as four

Table III. Cumulative Effects of Stage Leaching

Test	85	70	69	68	77	75	76
Time, hr	2	4	4	4	3	3	3
Temperature, °C	160	91	93	95	90	90	90
NaOH, g per liter	403.0	398.4	398.4	398.4	248.7	248.7	248.7
Pulp dilution	2 to 1	2 to 1	3 to 1	6 to 1	2 to 1	3 to 1	6 to 1
<i>First stage</i>							
Extraction, pct:							
SiO ₂	64.5	47.7	50.3	54.7	48.0	50.1	55.3
Al ₂ O ₃	7.7		0	1.8	0	3.6	26.4
Weight loss, pct	14.4	10.9	12.3	16.5	11.8	12.8	12.9
Residue, pct:							
Mn	36.15	36.0	35.8	36.4	35.54	35.94	36.38
SiO ₂	8.70	12.3	11.88	11.36	12.34	12.00	10.76
Al ₂ O ₃	3.36		4.34	3.66	3.55	3.45	2.64
<i>Second stage</i>							
Extraction, pct:							
SiO ₂	68.3	55.8	57.7	63.0	54.2	57.1	59.2
Al ₂ O ₃	15.7	2.1	2.7	13.8	4.7	18.1	31.2
Weight loss, pct	17.0	14.5	15.5	20.5	13.8	15.7	16.3
Residue, pct:							
Mn	36.30	36.6	37.0	37.6	36.33	36.73	37.57
SiO ₂	8.02	10.86	10.48	9.76	11.14	10.66	10.22
Al ₂ O ₃	3.16	3.58	3.58	3.38	3.45	3.03	2.56
<i>Third stage</i>							
Extraction, pct:							
SiO ₂	75.0	63.3	63.1	66.8	56.9	60.5	61.4
Al ₂ O ₃	19.1	15.2	6.4	33.5	16.4	30.1	29.9
Weight loss, pct	21.2	16.4	17.7	22.0	15.3	17.7	16.4
Residue, pct:							
Mn	38.1	38.5	37.6	38.7	36.98	37.62	38.21
SiO ₂	6.64	9.22	9.40	8.92	10.68	10.06	9.68
Al ₂ O ₃	3.20	3.18	3.54	2.66	3.08	2.65	2.62
<i>Fourth stage</i>							
Extraction, pct:							
SiO ₂		63.2	67.1	70.5	60.1	64.1	65.4
Al ₂ O ₃		14.6	31.7	48.4	22.4	42.6	33.2
Weight loss, pct		18.0	19.9	23.6	17.0	18.8	17.9
Residue, pct:							
Mn		37.52	38.51	39.40	37.27	37.87	38.81
SiO ₂		9.44	8.60	8.10	10.08	9.28	8.84
Al ₂ O ₃		3.27	2.64	2.10	2.92	2.21	2.54
Ignited at 1200°C:							
Ignition loss, pct		13.83	14.20	13.95	13.33	14.20	14.87
Mn, pct		43.41	45.08	45.86	43.13	44.12	45.33

stages of leaching at 90°C. The concentration of NaOH, within the range tested, had only minor effects on the extraction of SiO₂.

It is difficult to explain the data for the extraction of alumina; although significant increases in extraction were obtained, they did not follow a regular pattern. It appears that increased extraction occurred after the major portion of the soluble silica had been removed in earlier stages of leaching. This may be related in some manner to the low silica content of the liquors in later stages.

Optimum conditions for leaching appear to be somewhat as follows. Caustic concentration should be in the range of 243.8 to 398.4 g per liter NaOH. Lower concentrations were found to be less effective. A pulp dilution of 6 to 1 was most effective, but because large volumes of liquor must be handled, pulp dilutions in the range of 2 to 1 to 3 to 1 may be more desirable. A leaching time in the range of 2 to 3 hr was effective, and the choice of temperature for leaching lies between leaching at atmospheric pressure and pressure digestion.

Pulps for study of settling rates were prepared by leaching in the above manner. Initial settling rates under several sets of conditions were fairly consistent and varied in the range of 0.41 to 0.59 ft per hr. Corresponding final settling rates varied in the range of 0.08 to 0.25 ft per hr, and the thickened pulp contained 45.8 to 55.2 pct solids. Area requirements varied in the range of 10 to 15 sq ft per ton-day of dry solids. Concentration of NaOH, pulp density, and temperature of pulp when settling had little effect beyond the variations noted above.

Filtration studies were made with a laboratory Oliver United Filter leaf with an area of 0.1 sq ft, using Vinyon 101N for the filter cloth. Pulps were prepared by leaching in the manner described above. Data from these tests were irregular in trend except for showing improved filtration at higher temperatures for the slurry and with increased pulp density. For a pulp dilution of 1 to 1 the area required to filter one ton of dry solids in 24 hr was 10.6 sq ft for filtering at a slurry temperature of 25°C. When filtering was done at a temperature of 65°C the area required was reduced to 9.2 sq ft. When slurry having a pulp dilution of 6 to 1 was filtered, at the same temperatures as noted, the corresponding area requirements were 59.2 and 31.8 sq ft. These values show the irregular nature of filtration data as obtained from this filter leaf. The concentration of NaOH in the slurry did not show any pronounced effects, but the blinding of the filter cloth with slimes very markedly slowed the filtration rate.

Filtration on a Buechner filter, while not evaluated quantitatively, showed no unusual difficulty. In some instances where leach liquors had been allowed to cool to room temperatures filtration was slow. This was also the case for the denser pulps where the liquors contained a high concentration of

silica. On cooling, these tended to become viscous and to crystallize out sodium silicate, which hinders filtration. The crystals of sodium silicate redissolve readily on heating the liquor to about 60°C.

Washing a filter cake ¼-in. thick with hot water equivalent to 1.5 lb of water per lb of dry solids produced a cake containing 0.21 pct NaOH or 4.2 lb of water-soluble NaOH per ton of dry solids. But along with this, an additional quantity of NaOH appears to be tied to the solids by stronger bonds, for

Table IV. Leaching Charleston Hill National Mines Ore

Test	Pulp Dilution	NaOH, G Per Liter	Temperature of Leach, °C	SiO ₂ Extracted, Pct	Weight Loss, Pct	Tailings Analysis, Pct	
						Mn	SiO ₂
103A	3 to 1	387.4	91	8.0	2.3	31.46	44.8
103B	3 to 1	401.9	91	8.5	3.6	31.46	45.2
106A	2 to 1	397.0	120	27.2	11.4	34.47	39.12
106B	2 to 1	397.0	125	44.6	20.5	38.09	33.04
110A	2 to 1	380.8	140	61.8	29.9	42.69	25.96
110B	2 to 1	372.0	145	80.3	40.0	49.50	15.60
109A	3 to 1	396.0	160	85.0	41.1	51.90	12.10
109B	3 to 1	396.0	160	89.8	43.6	53.30	8.60
104A	2 to 1	245.1	160	72.1	34.7	46.49	20.36
104B	2 to 1	245.1	160	86.0	39.0	51.90	10.90

The A and B designations are first- and second-stage leaches, respectively.

the residues contained a total of 1.64 pct Na₂O, whereas the initial solids before leaching contained only 0.17 pct. The difference between these values represents NaOH permanently removed from the leach liquor by the cake. This amounts to 38 lb of NaOH per ton of dry solids. Prorating this value to the original concentrates reveals a loss in leaching of 32 lb NaOH per ton of concentrates treated. Of this loss, less than 4 lb can be recovered by additional washing.

Charleston Hill National Mines, Winnemucca, Nev., Experiment: Two samples of this ore were tested, and both consisted almost entirely of cryptocrystalline or metacolloidal silica carrying the manganese minerals. No quartz was observed in either sample. Table IV presents the results for the leaching of one of these samples. These data illustrate very vividly the effects of temperature on the leaching of silica from this ore, with consequent increase of manganese values. The head sample for these tests contained 30.83 pct manganese and 47.60 pct SiO₂. At 90°C only 8.5 pct of the SiO₂ present was removed by leaching, while at 160°C, 89.8 pct was extracted; respective manganese values were 31.46 and 53.3 pct. Tests 103 and 106 were run at atmospheric pressures, and the temperature for the latter test was near the boiling point for the slurry. Tests for the higher temperatures were run in a pressure digester. Values for the silica extraction and for weight loss are cumulative effects for the two stages of leaching. The other sample contained 18.5 pct Mn, 67.1 pct SiO₂, and 0.6 pct Al₂O₃. Leach-

Table V. Leaching of Other Ores

Source of Ore Sample	Head Analysis, Pct				Leach Tail Analysis, Pct				SiO ₂ Extracted, Pct
	Mn	SiO ₂	Al ₂ O ₃	Fe	Mn	SiO ₂	Al ₂ O ₃	Fe	
Aroostook, Maine	13.5	23.7	8.5	21.0	14.8	13.5	9.1	25.9	52.0
Rhodonite	29.3	48.7			35.6	38.0			
Three Kids manganese ore, Henderson, Nev.									
Stockpile No. 2	24.1	31.1	6.3		28.2	18.3	6.7		50.5
Rougher concentrate	34.2	16.2			36.2	8.9			52.0
Cleaner concentrate	38.4	8.8			39.9	5.7			37.4

ing produced a concentrate carrying 54.1 pct Mn and 11.9 pct SiO₂ where 90.8 pct of the SiO₂ was removed.

Other Ores Tested: Several other ores were tested in a similar manner with results as recorded in Table V. These data demonstrate the degree to which other types of ores will respond to leaching in NaOH solutions. In correlating the degree of leaching with the types of ores or concentrates leached one observes that those samples containing cryptocrystalline silica respond most readily, while those that carry quartz and silicate minerals are more refractory to this method of treatment for upgrading manganese. The leaching of rhodonite appears to have removed mostly the silica in excess of that required for the mineral. The Three Kids Stockpile No. 2 sample contained quartz, chalcedony, and opal as major gangue constituents; calcite, gypsum, clay, barite, and feldspars as intermediate constituents; and muscovite, biotite, and epidote as minor gangue constituents.

Additional points of interest were determined during leaching of Three Kids ore. Here a wider range of NaOH concentrations was used for leaching, and for conditions similar to those shown in Table II the following extractions of silica were obtained:

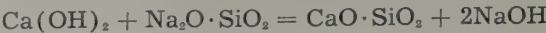
NaOH, G Per Liter	SiO ₂ Extracted, Pct
110.9	27.9
174.7	34.5
243.8	40.9
318.6	43.1

These data demonstrate the optimum concentration of NaOH required to obtain satisfactory extraction of silica. Further, the Stockpile No. 2 sample carried 1.8 pct lead. This lead could not be extracted to any appreciable degree by the leaching procedure employed.

Regenerating Caustic Leach Liquors

The leaching of manganese ores or flotation concentrates with caustic soda produces a liquor containing essentially a sodium silicate. As the silica concentration in the leach liquors increases, there is a decrease in the efficiency of leaching, and it becomes necessary to regenerate the NaOH in spent leach liquors to restore their leaching efficiency and recover NaOH for further leaching. Regeneration, then, becomes an important secondary operation for the process. Spent leach liquors were regenerated

by adding lime to precipitate calcium silicate and free the NaOH, according to the general equation:



In using this reaction for regeneration, one must know the conditions under which it will function most efficiently. The following discussion and data outline those conditions.

Experimental Procedure: Spent caustic leach liquors from leaching operations were made up into slurries by adding appropriate quantities of dry lime, added at a definite mol ratio of CaO to SiO₂, based on the SiO₂ present in the liquor sample. The slurry was then placed in a water bath to maintain a constant temperature during the reaction period and stirred vigorously at regular intervals. After an appropriate reaction time, the slurry was filtered on a Buechner filter at temperatures corresponding to those used for the reaction but not greater than 60° to 75°C. Filtrates were analyzed for total NaOH by titrating with a standard acid, and no distinction could be made between free caustic and that combined with silica. The silica remaining in the liquor after regeneration was used as a measure for its efficiency. Relating this value to other variables outlined the optimum conditions required for regeneration.

Experimental Results and Discussion: The effective regeneration of calcium silicate was obtained by adding dry fresh-burned lime to the spent leach liquor in a mol ratio of lime to silica of 1.0 and reacting this mixture for a period of 2 hr at a temperature above 60°C. Lower temperatures and shorter periods of time were less effective in producing a calcium silicate that was readily handled. Reaction periods in excess of 2 hr and mol ratios in excess of 1.0 did not materially improve the process of regeneration. Fig. 1 illustrates the effects of reaction time vs the grams per liter of silica remaining after regeneration. The shift in location of these curves is due to differences in NaOH concentration in solutions from which precipitation was made. Fig. 2 explains this shift. Nearly identical curves in shape and location were obtained for mol ratios of lime to silica vs grams per liter of silica remaining. The initial silica concentration in the spent leach liquors had no effect on the progress of regeneration, except that liquors containing 75 g of silica per liter or more could not be regenerated because such slurries became thick muds that would not flow readily. Those solutions containing 50 g of silica per liter could be handled easily.

The effects of NaOH concentration on regeneration were very pronounced and in these tests the concentration was varied from 200 to more than 400 g of NaOH per liter. In Fig. 2 the grams of silica per liter remaining after regeneration are plotted in comparison with the concentration of NaOH in the solution from which precipitation was made. In every case where the mol ratio of lime to silica was 1.0 or more, the final value for the silica remaining in the leach liquor fell near Curve A. It suggests that equilibrium conditions of some kind were established between the reactants and the products. It follows, therefore, that for liquors of a given NaOH concentration containing less silica than a value determined by Curve A, little silica will be precipitated by the lime and, further, that for higher silica values only that portion above Curve A will be precipitated.

A mol ratio of lime to silica of 0.57 was used for precipitation for obtaining Curve B in Fig. 2. This

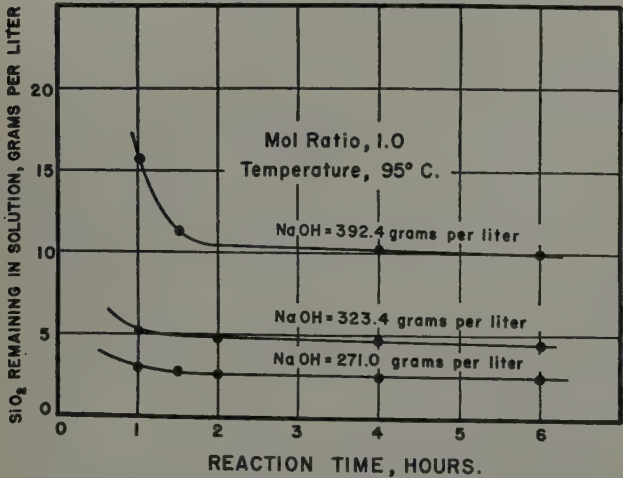


Fig. 1—Effect of reaction time on the regeneration of NaOH, when a lime-to-silica mol ratio of 1.0 is used.

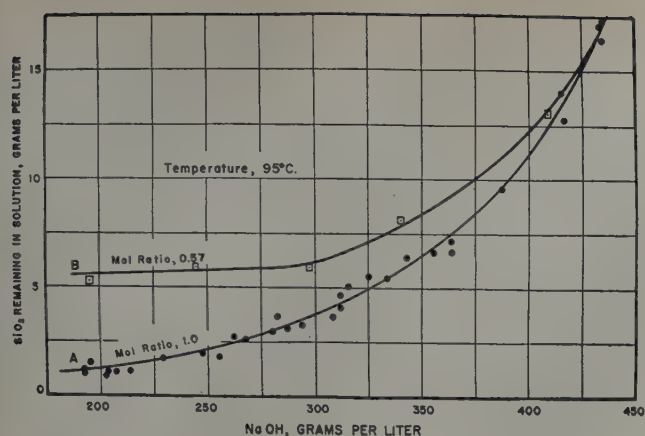


Fig. 2—Effect of NaOH concentration on regeneration, when lime-to-silica mol ratios of 0.57 and 1.0 are used.

curve shows that for the lower concentrations of NaOH an insufficient quantity of lime had been added to meet the demands required by Curve A and, therefore, that portion of Curve B is flat. However, as the concentration of NaOH increases, this quantity of lime added approaches and finally equals or exceeds the demand for Curve A. Consequently the two curves approach each other and finally merge. The silica left in solution representing the difference between Curve A and Curve B can be precipitated by the addition of an appropriate quantity of lime.

The slimy nature of calcium silicate precipitates precludes the use of thickeners for separating the solids from the liquor except for very dilute slurries. The heavier slurries must be filtered to make the separation, and filtration on a Buechner filter proceeded at a satisfactory rate for solutions containing 350 g of NaOH per liter or less. For the more concentrated solutions the filter rate decreased rapidly as the NaOH concentration increased. The unwashed filter cakes contained about 65 pct moisture.

Washing these cakes with about 3.5 volume displacements of hot water reduced the NaOH retained to about 5 pct of the weight of the dry solids. This is an average value for retained NaOH that varied somewhat with the concentrations of NaOH in the initial liquor, cake thickness, cracking of the cake, and the physical nature of the solids. Extreme values for retained NaOH varied from 1 to 10 pct and the wet washed cake retained about 80 pct moisture.

It was found that fresh-burned lime produced excellent results for regeneration when it was slaked in the spent leach liquor to be regenerated. Hydrated lime added in the same manner did not produce comparable results. The following analysis of a calcium silicate precipitate was typical of those obtained:

Item	Percent
CaO	38.0
SiO ₂	38.5
Na ₂ O	3.6
Mol ratio CaO to SiO ₂	1.055

From Table III it may be anticipated that about 280 lb of silica extracted from a ton of Artillery Peak flotation concentrates would have to be precipitated. This would require about 280 lb of lime and there would be about 727 lb of dry precipitate produced which would carry about 33 lb of NaOH. Combining this value with the loss occurring in leaching, there would be some 65 lb of NaOH lost in treating

one ton of original flotation concentrates exclusive of all other losses.

Seeding Out Sodium Silicate: Spent caustic leach liquors will drop out crystals of sodium silicate on cooling and standing. This crystallization may be speeded by cooling the liquors to 20°C or less and adding a few small crystals of sodium silicate. To test the efficiency of this method for the removal of SiO₂, a liquor containing 286.7 g NaOH and 45.93 g SiO₂ per liter was cooled to 20°C, and a few small crystals of sodium silicate were added. In 1 hr, the entire volume was solid with loosely packed crystals. The mass was gently stirred and allowed to settle for 17 hr before the crystals were separated from the liquor. The new liquor had lost 19.0 pct of its original volume and now contained 268.4 g NaOH and 11.77 g SiO₂ per liter. Crystallization of sodium silicate had removed 24.1 pct of the NaOH and 89.2 pct of the SiO₂ from the original leach liquor. The mol ratio of Na₂O to SiO₂ in the crystals recovered was 1.43.

Build-up of Impurities in Leach Solutions: Certain minerals often found associated with manganese will react with NaOH to produce sodium compounds that will not readily regenerate NaOH. These compounds will in time build up and render the solution inert for leaching silica. Gypsum is a common associate in manganese ores, and it will convert NaOH into Na₂SO₄. Carbonates or carbon dioxide from a number of sources will produce Na₂CO₃. Potash, or K₂O, probably from the clay minerals, also accumulates in the leach liquors. Alumina showed no tendency to accumulate in the leach liquor. The rate at which these impurities accumulate and, therefore, the rate of decrease of leaching efficiency of NaOH was not determined. Undoubtedly, some system of periodic removal of these and other impurities must be incorporated in the flow sheet.

Summary

A study was made to determine the effectiveness of leaching manganese ores or flotation concentrates with NaOH solutions to remove soluble silica and silicate minerals. It was found that some types of ores or concentrates could be upgraded appreciably, producing concentrates carrying more than 40 pct manganese, and that the spent leach liquors could be regenerated for re-use in further leaching by treatment with fresh-burned lime to precipitate the silica as calcium silicate.

Acknowledgments

Research and development work on this process for upgrading manganese ores was conducted by the Bureau of Mines Manganese Research Section at the Electrometallurgical Experiment Station and Pilot Plants, Boulder City, Nev. The author is grateful for the many helpful suggestions and constructive criticism offered by Rex R. Lloyd, Chief of the Boulder City Station, and by James H. Jacobs, formerly engineer in charge of the Manganese Research Section. He wishes also to thank Charles T. Baroch, senior metallurgist, for his critical review of the manuscript. The author further acknowledges the work of Thomas E. Hill, Oliver O. Leone, and William H. Curry in conducting much of the experimentation, and the work of P. R. Perry and his assistants in performing the analyses required for this study. To all these people and to many others not mentioned who assisted in this work indirectly the author expresses his appreciation.

Double-Bond Reactivity of Oleic Acid During Flotation

by A. M. Gaudin and R. E. Cole

OLEIC acid, a standard flotation reagent, has generally been preferred to other fatty acids. Because oleic acid differs from saturated fatty acids by the presence of one carbon-to-carbon double bond and because the flotation operation is one in which extremely intimate exposure of the reagent to oxygen is obtained, it appeared possible that the value of oleic acid as a flotation collector was related to the oxidizability of its double bond.

To test this hypothesis, the floated mineral, fluorite, was leached with a solvent capable of extracting the adsorbed reagent and the extract was analyzed.¹ Results show that oleic acid is not oxidized during flotation, although some oxidation can be obtained if the flotation operation is repeated many times and if oxygen in place of air is employed as the gas during flotation. Incidentally, it was found that linoleic acid which has two nonconjugated double bonds is not affected appreciably more than oleic acid. Linolenic acid which has three nonconjugated double bonds is measurably altered even during one flotation step, using air.

The fatty acids used, from the Hörmel Foundation, were of the highest purity. The quantity of fatty acids employed in each test was sufficient to float the mineral, but less than the theoretical amount required to form a monolayer at the mineral surface. The purpose of this limitation was to eliminate droplets of oil smeared on the adsorbate-covered mineral surface. The mineral was dried in vacuum at about 60°C. Effective leaching was obtained by use of a solution of 1 pct hydrochloric acid in absolute alcohol. The extract was separated from the solvent by distillation, with final recovery of the extract in carbon tetrachloride.

The identity of the extract was tested by a combination of analytical procedures including infra-red spectroscopy, saponification number, iodine number, and index of refraction. At the outset, great hopes for accurate quantification of oleic acid oxidation had been entertained in regard to the method of infra-red spectroscopy. Unfortunately, because the double bond in the oleic acid is so sym-

metrically located at the center of the molecule that it provides very weak absorption bands, accurate quantitative analyses were not obtained. Nevertheless, infra-red spectrograms suggested the absence of double-bond oxidation during flotation.

Table I. Iodine Values of Extracted Adsorbates

Test No.	Fatty Acid Collector Used	Extract Iodine Values		
		Observed	Theoretical*	Calculated**
A	Oleic lot 5	80.62	81.75	81.21
B	Oleic lot 5	80.67	81.75	81.21
C	Oleic lot 4	81.28	81.75	82.34
D	Oleic lot 4	81.21	81.75	82.34
E	Linoleic	164.37	164.57	162.81
F	Linoleic	164.14	164.57	162.81
G	Linolenic	231.62	248.48	247.56
H	Linolenic	230.85	248.48	247.56

* The theoretical values listed are those of pure ethyl esters of the fatty acid collectors.

** Iodine values of the ethyl esters as calculated from the determined iodine values of the unused fatty acid collectors.

Iodine number, semi-micro method, provided a critical test of the extent to which fatty acids oxidized during flotation, see Table I. The index of refraction was used in a qualitative rather than quantitative fashion, and its findings corroborated those obtained by the iodine-number method.

It is concluded, in the case of fluorite and C-18 fatty acids having one or two nonconjugated double bonds, that there is practically no change of the fatty acid molecule or ion during the flotation operation. It would appear, therefore, that the extraordinary utility of these reagents is related to some property of the compounds other than the oxidizability of their double bonds.

Acknowledgment

The research reported herewith was sponsored by the Research Division, U. S. Atomic Energy Commission, and for this help the authors express their appreciation.

Reference

¹ R. E. Cole: "Double-Bond Reactivity of Oleic Acid During Flotation." Thesis submitted in partial fulfillment of the requirements for the degree of M.S. at the Massachusetts Institute of Technology.

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TN 1448. Manuscript, October 21, 1952. Los Angeles Meeting, February 1953.

Uranium Mineralization in the Sunshine Mine, Idaho

by Paul F. Kerr and Raymond F. Robinson

Uranium mineralization occurs in the footwall of the Sunshine vein from the 2900 to the 3700 level. Veinlets of uraninite associated with pyrite and jasper have been so extensively divided and recemented that units more than a few feet in length are seldom observed. The wall rock is St. Regis quartzite of the Belt series. The age of the uraninite, on the basis of isotopic analyses, is 750 ± 50 , which agrees with geological data suggesting that phases of the Sunshine mineralization are pre-Cambrian.

THE Sunshine mine in the Coeur d'Alene district, Idaho, is well known for its silver-bearing veins but prior to the summer of 1949 had not been recognized as a possible source of uranium. At that time, during a geiger counter reconnaissance by T. E. Gillingham, R. F. Robinson, and E. E. Thurlow, high radioactivity was noted and radioactive specimens were collected from the footwall of the Sunshine vein.¹ The detection led to the identification of uraninite-bearing veins, since explored jointly by the Atomic Energy Commission and the Sunshine Mining Co.

After the occurrence was noted, the geology of the uranium deposit was studied by the Sunshine staff, and a laboratory examination of the ores was conducted at Columbia University. Several types of laboratory work were undertaken. Differential thermal curves were made of selected siderite samples and results from many more were secured through the work of Mitcham.² X-ray diffraction and X-ray fluorescence analyses were employed on uraninite, jasper, and siderite. Chemical analyses were made through the cooperation of the Division of Raw Materials of the Atomic Energy Commission.

General Geological Features

Several silver-bearing veins cut the overturned north limb of the Big Creek anticline as mapped by Shenon and McConnel,³ while the Osburn fault, a long-recognized regional feature about a mile away, marks the north boundary of the Silver Belt.

The Sunshine vein, Fig. 1, has a south dip more or less parallel to the 60° axial plane of the fold and cuts rocks of the Belt Series, starting with the Wallace formation near the surface, continuing downward through the St. Regis formation, and probably extending into the Revett quartzite which lies below the bottom or 3700-ft level.

The limb of the anticline is locally modified by secondary folds, one being prominently exposed in the uranian area along the Jewell crosscut near the Sunshine vein. Crumpling of the limb resulted from compression which formed the anticline and probably preceded the faults in which the vein deposits accumulated. Evidence of drag along these faults points to reverse movement in the uranium-bearing

area and elsewhere. This is true of major faults in the mine workings, and the majority of faults which can be mapped, as pointed out by Robinson.⁴

The St. Regis formation, as measured in the mine, appears to have an initial thickness of some 2000 ft, but the apparent thickness due to thickening during folding is some 3400 ft. Along the Sunshine vein the purple and green rocks characteristic of the Wallace formation in the nearby Military Gulch section⁵ (p. 37 of ref. 5) have been completely *bleached* because of introduced sericite. Hydrothermal solutions acting on the wall rock have substituted for the original color a pale greenish cast, although no pronounced mineralogical change has resulted, as Mitcham has observed.²

The silver and the uranium depositions appear to belong to distinct epochs resulting from several periods of emplacement. Likewise, multiple periods of deformation account for the faulting. Uraninite is generally associated with silicification, while silver mineralization accompanies carbonate veins. Rarely, uraninite may be found in a matrix of siderite. Ordinarily uraninite formed prior to argentinian tetrahedrite. Where clusters of veins form a stockwork, uraninite-jasper veins often favor one trend while tetrahedrite-siderite veins favor another.

During deformation, brecciation of the St. Regis quartzite provided openings between broken rock fragments for precipitation from vein-forming solutions. Fractures due to major breaks were filled during the first stages of vein formation, while later deformation displaced the first veins and provided new channels along which further mineralizing solutions proceeded.

The uraninite veins, as the first formed, have suffered fracturing, displacement, and segmentation. Uranian vein segments uncut by faults and more than a few feet in length are rare or non-existent. Siderite veins are more massive and often extend without a break for tens and even hundreds of feet. In general they show much less segmentation. While the siderite is usually later, there is an overlap in the periods of deposition, some earlier siderite veins being extensively segmented in much the same way uraninite veins have been broken.

Vein silica is more extensively distributed than the uranium and iron mineralization it carries. Along the vein course concentrations of uraninite frequently fade away and barren white quartz continues, the transition often occurring within a few feet along strike or down dip. An example appears on the 3700-ft level where a uraninite vein, see Fig. 2a,

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Discussion on this paper, TP 35321, may be sent (2 copies) to AIME before July 31, 1953. Manuscript, May 15, 1952. Revised, Dec. 4, 1952. New York Meeting, February 1952.

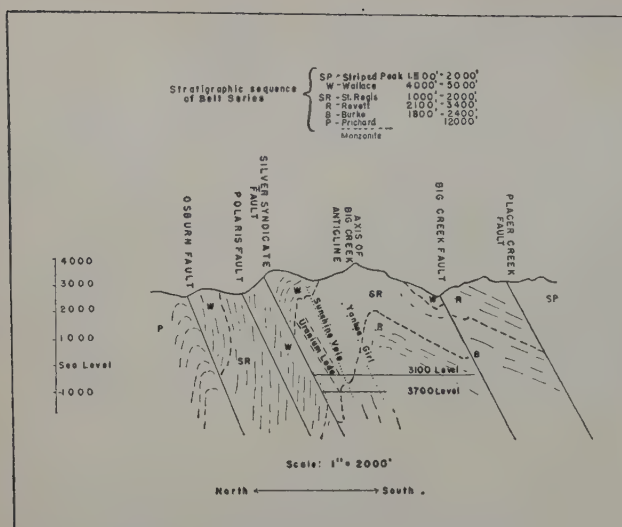


Fig. 1—General structure of the Sunshine mine, shown here in section.

changes within 20 ft to a barren ribbon quartz vein, see Fig. 2b.

Deformation

Uraninite veinlets form clusters near the axis of an anticlinal fold on the north flank of the main anticline, Fig. 3. On the 3100 level west, the crest of the fold has a westward pitch of 15° to 20°, while 200 ft above, the same axis has a westerly pitch of 25°. Below, on the 3700 level, a single uranium vein lies a short distance north of the axis. Here the flank of the fold is slightly overturned to the north. Veins are crumpled in places with post-vein deformation, Figs. 2a and 2b, and are frequently cut by low angle cross faults, or *flat faults*.

The relationship of two silver veins, the Sunshine vein and the 06 vein, to the uranium-bearing zone, is shown in Fig. 3. The strongest uranium mineralization occurs near the west end of the drift but it is noticeably discontinuous, Fig. 4. Veins exposed in drifts and crosscuts may be missing in drill holes a short distance away along the vein strike, while occurrences in drill holes may not continue into nearby workings. On the other hand, the Sunshine

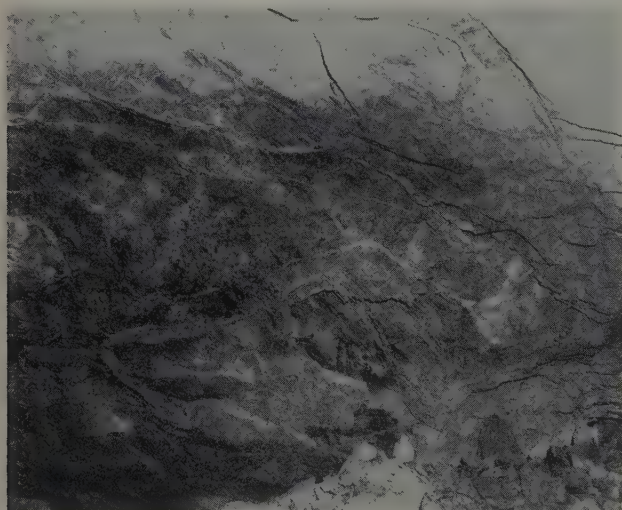


Fig. 2a—Uraninite vein, 3700 level. Photograph taken near the Jewell crosscut, looking southeast, shows a surface freshly exposed after blasting. Uranium-bearing zone is outlined with white chalk.

and the 06 veins which cut across the uranium zone are continuous throughout the length of the drift.

Low Angle Faults

Flat faults cut the uranium veins and also limit deposition. The fault planes are often irregular, many being folded. Well shown in a stope, 16W, a prominent zone of uraninite stringers follows fracture cleavage westward until cut by a fault plane inclined at less than 10° to 25°. Earlier uraninite vein strands are terminated by this fault, but some later uraninite veinlets follow the fault plane itself.

Flat faults may follow the bedding planes of the St. Regis formation where the strata are inclined at less than 25°S, or they may cut the stratification at a small angle. In the mine area slippage along bedding planes is noticeable, particularly near the local folds. The displacement of uraninite veinlets by a flat fault may be limited to a few feet and in places reduced to fracturing without displacement.

Fig. 5 illustrates a flat fault. Both gouge and quartz introduced along the fractures are impregnated with radioactive particles. Pyrite and arsenopyrite occur in the shear zones. Quartz is also present and changed in places to jasper. The quartzite both above and below the fault is red and radioactive, although more deeply colored and mineralized below. Early siderite-pyrite veins show drag and are truncated by the fault. Arsenopyrite and pyrite of the fault zone are later than the early pyrite of the steep veins. The steep veins taper away from the fault and may have derived their filling from the fault channel at an earlier stage. Thus the flat fault represents more than one generation both of movement and mineralization.

A group of flat faults is illustrated in Fig. 6. These are filled with gouge, curved, and split. Both the St. Regis quartzite and earlier groups of siderite veins are displaced. Ordinarily these faults merely cut veins, but later white quartz at the bottom of the section cuts both early veins and gouge.

Fig. 7 shows a flat fault cutting a system of veins. Movement has occurred along an irregular surface, the two walls being separated by gouge of varying thickness. The patterns of the two walls fail to match, indicating substantial displacement.

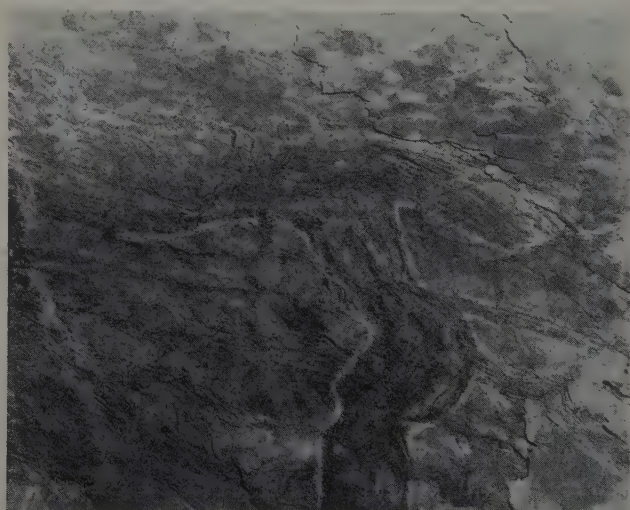


Fig. 2b—Ribbon vein, 3700 level, terminating uranium mineralization. A photograph of a later state in the development of the uraninite vein, showing barren quartz stringers in a crenulated pattern. The boundaries of the zone are indicated by chalk.

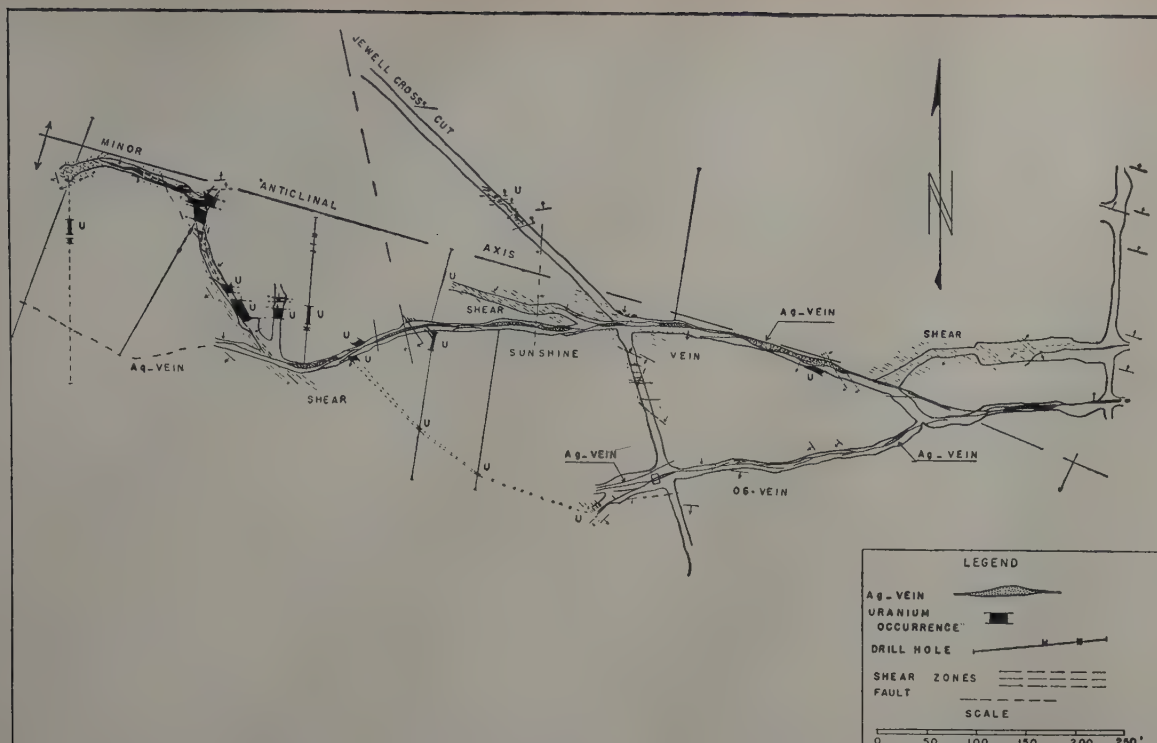


Fig. 3—Uranium and silver veins, 3100 level. The uranium zone intersects with the silver-bearing Sunshine and O6 veins in the vicinity of a minor anticlinal axis.

Above the fault a low grade silver-siderite vein and a light red jasper vein occur a few feet apart. The low grade silver-siderite vein has been cut by the fault, while later white quartz merges with and follows the fault line. About 10 ft below the fault a stringer of red jasper and pyrite merges into arsenopyrite schist. The sequence seems to have been: 1—arsenopyrite vein wall impregnation; 2—pyrite and jasper veins, uranium-bearing; 3—silver-siderite veins; 4—flat cross faulting; 5—white quartz cross veins.

Evidently near the flat faults the flow of solutions carrying uranium continued for a considerable time,

Fig. 7. Flat faults are mineralized, Fig. 5, with the possibility that mineralization and movement were partly concurrent.

Shear Zones

Fracture cleavage in the quartzite of the Silver Belt has been described by Shenon and McConnel.⁶ Along the 3100 level the angle with the bedding is normally 30° to 45° and shear zones appear to be more highly developed on the flanks of minor folds. The shear zones have provided openings along which both silicification and uranium mineralization have taken place.

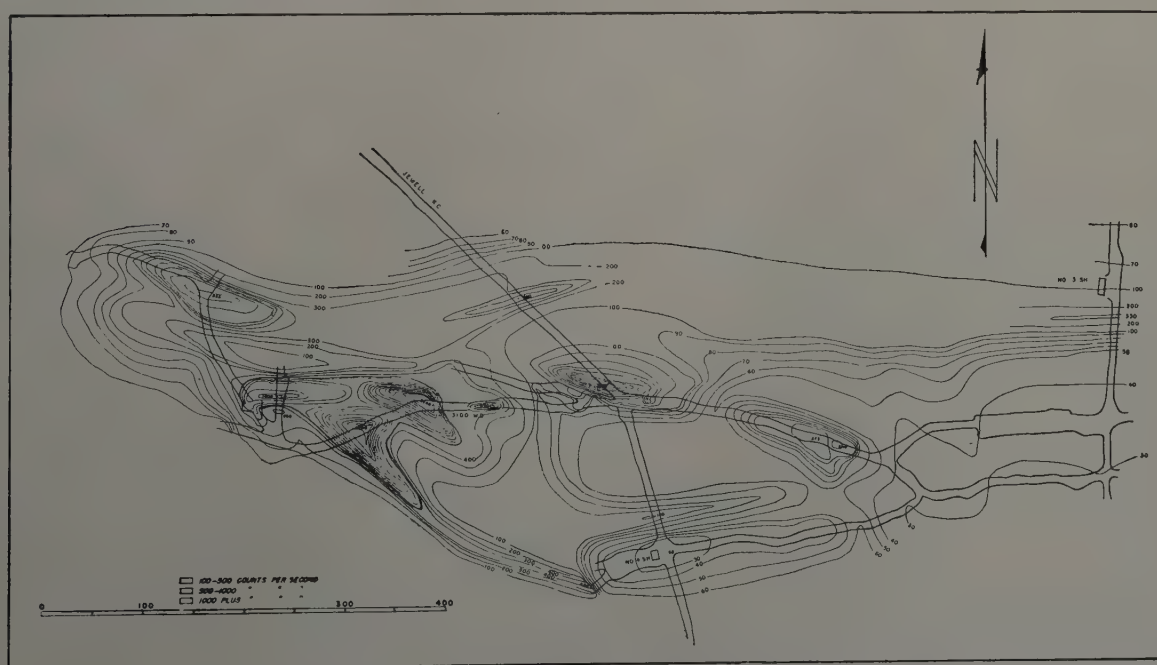


Fig. 4—Radioactivity along 3100 level. The diagram is based upon observations along drifts and crosscuts, and also takes into consideration comparative radiometric data from drill holes.

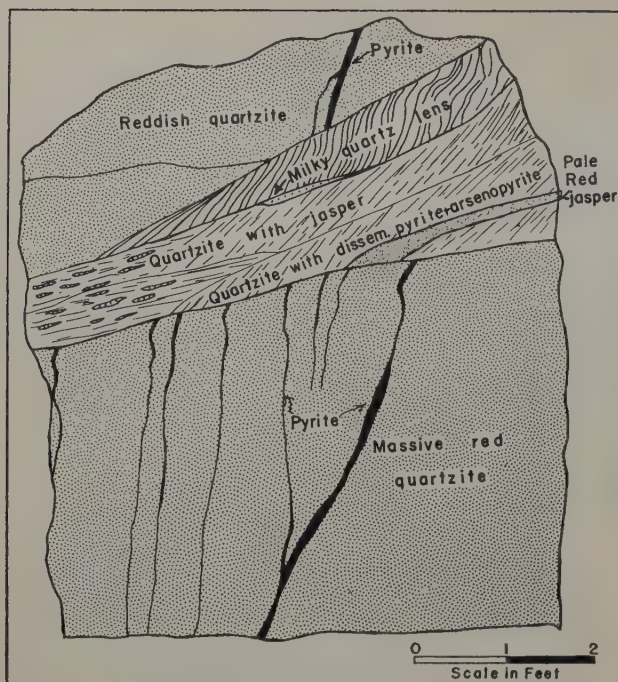


Fig. 5—Shear zone along a flat fault at the end of the north-west drift, 3100 level west, in the footwall area of the Sunshine vein.

Shear zones which may carry uranium mineralization are distributed throughout a total spread of some 200 ft wide. While they have offered favorable loci for the emplacement of uranium-bearing veinlets, the actual precipitation is erratic and considerable intervening non-mineralized areas exist. Fig. 8 illustrates shearing along a vein.

Fig. 9 illustrates an exposure showing several high grade uraninite stringers cut by a flat fault. The quartzite is intensely sheared and small fissure fillings containing uraninite are dispersed along fracture planes. Although cross-fractured, the veins have been only slightly displaced. In places the quartzite has

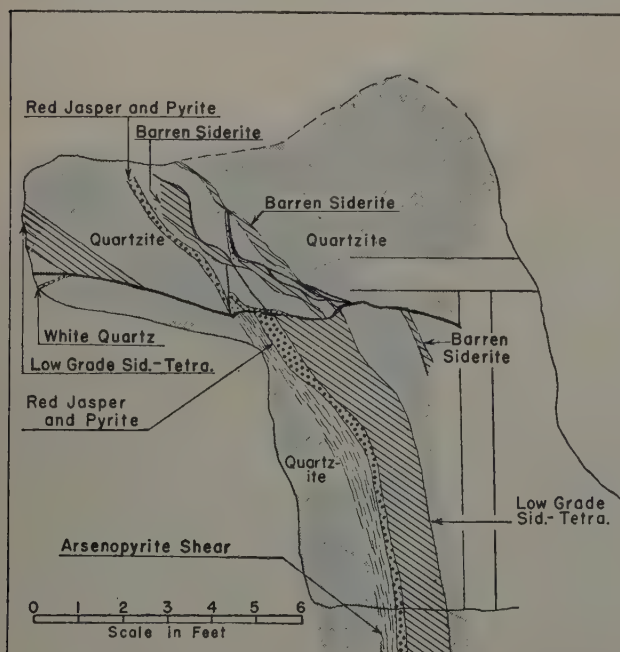


Fig. 7—The section shows an irregular flat fault displacing a low grade tetrahedrite-siderite vein and a shear zone containing arsenopyrite. East end 25 E. stope, 3250 level.

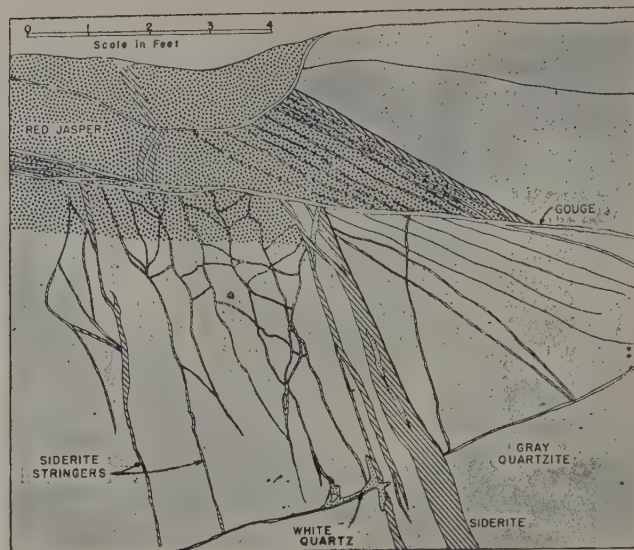


Fig. 6—Flat faults cutting the vein pattern.

been sheared and impregnated until it is locally an arsenopyrite schist. In a few places where arsenopyrite is abundant, orpiment has developed. Veinlet No. 1 in the diagram consists of red jasper, uraninite, and pyrite above a portion of a bedding fault opposite a siderite vein below. The thin veins, No. 2, consist of up to 3 to 4 in. of red jasper, uraninite, and pyrite both above and below a bedding fault. Pink quartz is present in the veinlets in the lowest part of the exposure.

The back of a stope in a plan view, Fig. 10, shows early diagonal cross fractures with uraninite veinlets and jasper having a diagonal trend along a cross shear pattern. Strike fractures followed with parallel movement furnish openings for siderite-tetrahedrite veins along the main shear. Late fracturing cuts all previous phases. Such field evidence indicates at least three main stages: early northwest shear, later east-west shear, and post mineral cross fracture.

Ribbon Structure

Banding with quartz or quartz and siderite veins forming a group of more or less parallel ribbons is a common feature in barren areas. Similar banding in uranium-bearing zones includes jasper and uraninite as well. A ribbon vein with associated uraninite is shown in Fig. 11. The ribbon structure consists of a succession of white quartz and siderite veins with border bands of pyrite-uraninite. A small, sinuous quartz vein containing galena and minor siderite was emplaced later. A similar ribbon structure mentioned above, Fig. 1b, is barren. The only feature which might give a clue to the proximity of nearby ore is dark gray quartzite with associated arsenopyrite.

Ribboned veins followed along the strike have led to uraninite-bearing zones, and in contrary manner, uraninite veins have merged into barren ribbon zones. The transitions may occur within a few feet. The sequence appears to be: main vein fracturing followed by deposition of arsenopyrite and pyrite; ribbon vein formation containing white quartz and siderite; late quartz filling which may be galena-bearing; and finally, late fracturing.

Age of Mineralization

Monzonitic rocks are exposed in an area north of Wallace, and dating of mineralization has previously

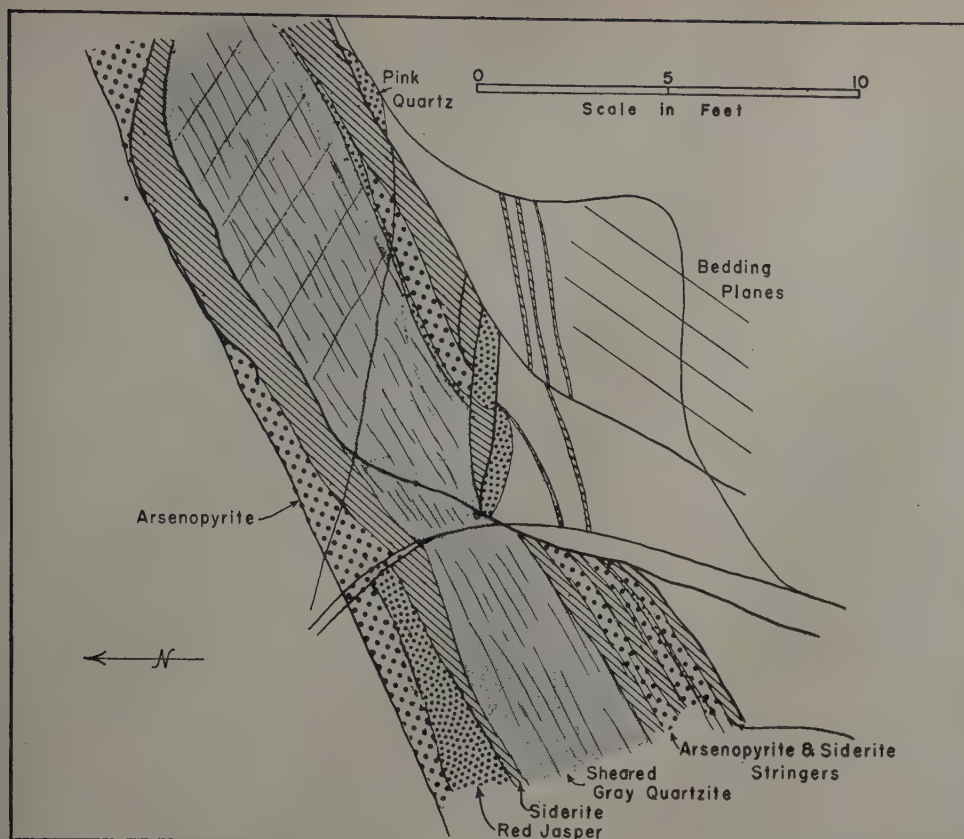


Fig. 8 — Section through 27 E. slope between 3250 and 3400 levels. Shows arsenopyrite vein borders, jasper carrying small amounts of uraninite, and siderite veins in an area heavily impregnated with arsenopyrite.

been based largely on the age of the intrusion. These rocks invade all members of the Belt Series except the Striped Peak. Anderson⁷ concluded that monzonitic rocks, zones of bleaching, and mineralization in the Coeur d'Alene district all show intimate dependence on a deep-seated differentiating magma.

On the basis of both chemical and lead isotope analyses, J. Laurence Kulp has computed the age of Sunshine uraninite from the 3100 level as 750 ± 50 million years.⁸ Further determinations will be required before the age of the uraninite may be considered other than tentative, but data now available favor a pre-Cambrian age.

Recently Anderson⁹ has pointed out the existence of five metallogenic epochs in Idaho, namely, 1—late

pre-Cambrian, 2—early Tertiary, 3—mid-Tertiary, 4—late Tertiary, and 5—Quaternary. He discredits the role of the Idaho batholith, late Mesozoic, as the most important mineralizer in the state.

Anderson considers that the monzonite of the Coeur d'Alene belongs to early Tertiary igneous activity and that the mineralization of the Silver Belt is related (p. 597 of ref. 9). The pre-Cambrian metallization is associated with the Purcell sills near the Canadian border. Copper deposits near Pocatello are also thought to be pre-Cambrian.

As the uranium mineralization is younger than the shears and fracturing in which it occurs, a pre-Cambrian age for the uraninite would date the folding as also pre-Cambrian. This is contrary to the

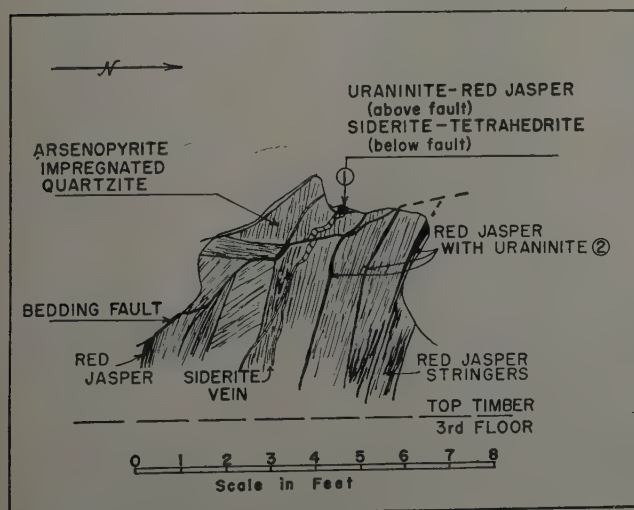


Fig. 9—Section shows sheared quartzite, impregnated with arsenopyrite and cut by uraninite veinlets. Although cross-fractured, the veins have been only slightly displaced.

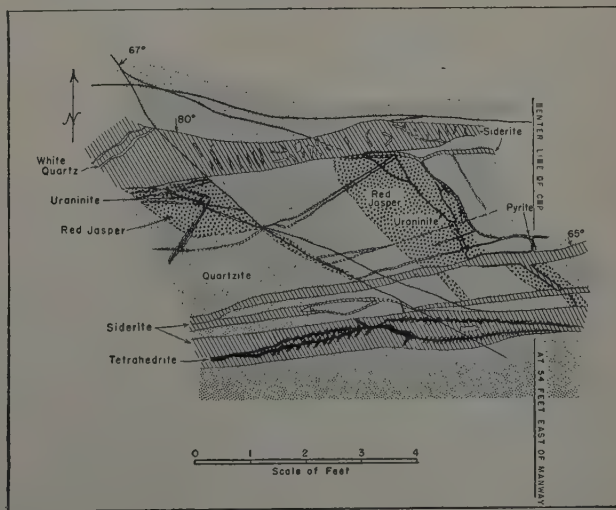


Fig. 10—Two intersecting zones of mineralization, 43 ft below 3100 level, west of Jewell crosscut. A north-west trending jasper zone and uraninite veinlets are cut by later east-west veins of siderite-tetrahedrite and siderite-quartz.

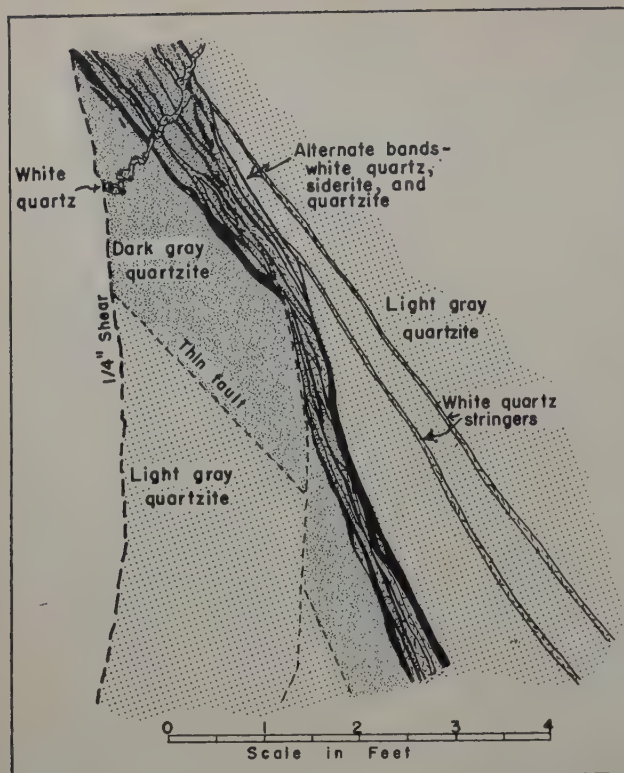


Fig. 11—A system of ribbon veins, 27 stope footwall crosscut, N-bearing 3400 level. The ribbon veins consist of alternate bands of white quartz and siderite, possibly with pyrite-uraninite.

customary interpretation which would place the folding in the Mesozoic. The segmentation of uraninite veinlets, the later infiltration of quartz around fragmented uraninite, the later reopening and sideritization of uraninite veinlets, and the truncation of uraninite veinlets by later siderite-tetrahedrite veins all point toward a later post-uraninite sulphide mineralization. It is not clear, however, whether the silver-bearing veins constitute a late phase of the pre-Cambrian mineralization or belong to one of the later epochs.

The Silver Syndicate fault furnishes evidence which may be significant. The gouge, the drag folds, and the shear have been replaced by the silver ore, and relict outlines are clearly preserved. The ore textures are unbroken by post-ore movement and indicate no major deformation since emplacement. Total absence of deformation along this structure since pre-Cambrian time seems unlikely. Yet later mineralization with an absence of more than minor differential movement since seems possible. At least one later epoch of mineralization probably occurred since pre-Cambrian. It seems likely that the Big Creek anticline as the major controlling structure dates to pre-Cambrian, but emplacement causing the heavy siderite veins and deformation responsible for certain major faults probably occurred later.

Types of Mineralization

Early Pyritic Veins

Massive pyritic veins with traces of fine disseminated uraninite occur near the top of the radioactive zone up to the 2900 level. Pyrite cements a vein breccia of angular quartzite. Later non-radioactive



Fig. 12a — (Left) Early uraninite-pyrite vein, 3100 level drift, area shown is 3x6 in. Quartzite with disseminated uraninite (gray) forms a halo bordering veinlets of early pyrite (white). Fig. 12b — (Right) Radiogram of the portion of early vein shown in Fig. 12a.

siderite-tetrahedrite veinlets cut the earlier pyritic vein fillings. Halos of black smoky alteration mark pyritic margins and fade outward into the wall rock. Included quartzite fragments are also smoky in places. Most of the nearby quartzite is reddish brown. Disseminated through the halos are minute crystals of pyrite and other sulphides with scattered specks of uraninite.

A disseminated uraninite halo is shown in Fig. 12a, where the bright areas are pyrite and the darker areas are reddish brown quartzite, some of which is quite smoky. At the extreme left of the specimen, 12a, is a vein of siderite, in which there are small patches of tetrahedrite (black). In the radiogram, Fig. 12b, the uraninite lies both in brown quartzite and disseminated through the pyrite. The radioactivity is greatest in quartzite bordering the pyrite and absent in siderite.

Fig. 13 shows a specimen from the 3100 level, with a pyrite vein cutting across grayish brown quartzite. Specks of finely disseminated uraninite occur within the pyrite, but a more intense halo of radioactivity grades into the rock on each side of the vein. Clusters of fine metallic particles lie further removed in the quartz. Such phenomena indicate that part of the uraninite was an early phase, probably slightly earlier than the pyrite veins.

In Fig. 14 faults cut a pyrite-uraninite vein into two segments. However, vein filling closely resembling the vein pyrite-uraninite follows one fault plane, indicating that part of the faulting existed prior to emplacement. The main emplacement, however, was pre-faulting. Siderite veins, on the other hand, are truncated or displaced, and most of the fault movement is post-siderite.

Fig. 13 (Below)—Pyrite vein with uraninite halo, 3100 level, crosscut at 75,000. Area shown is 3x4 in. The pyrite veinlet contains specks of uraninite and is bordered by a diffused halo containing fine uraninite particles in quartzite.

Fig. 15 (Right)—Early uraninite (black) cut by late siderite (light gray), 25 E. stope, 3250 level, area shown is 3x8 in.

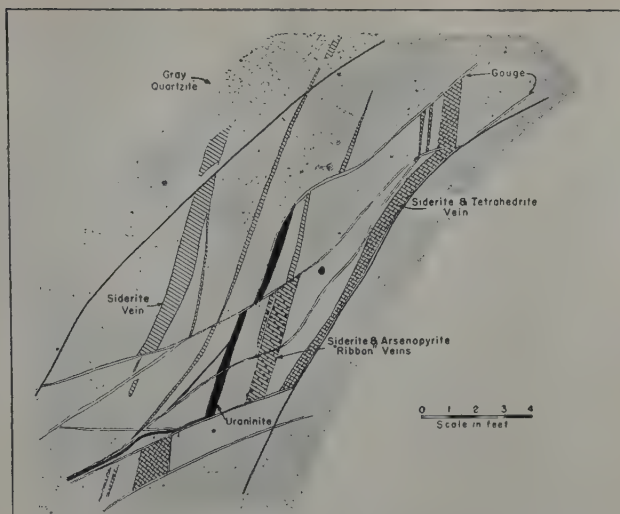
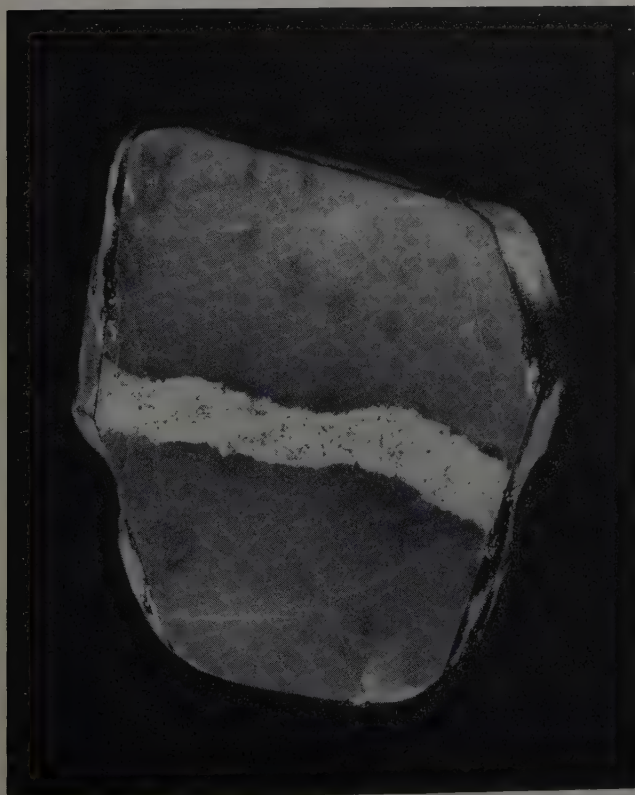


Fig. 14—A pyrite-uraninite vein is segmented in part by cross faulting and also follows a cross fracture.

Uraninite where most abundant occurs in veinlets consisting of intermixed red jasper, pyrite, and uraninite. Several types of associated pyrite are found. Finely crystalline pyrite occurs in at least two generations associated with uraninite. A more coarsely crystalline euhedral pyrite is usually of later origin.

Uraninite-Quartz-Pyrite Veins

Early uraninite may be cut by later silver-siderite mineralization, as shown by a specimen, Fig. 15, col-



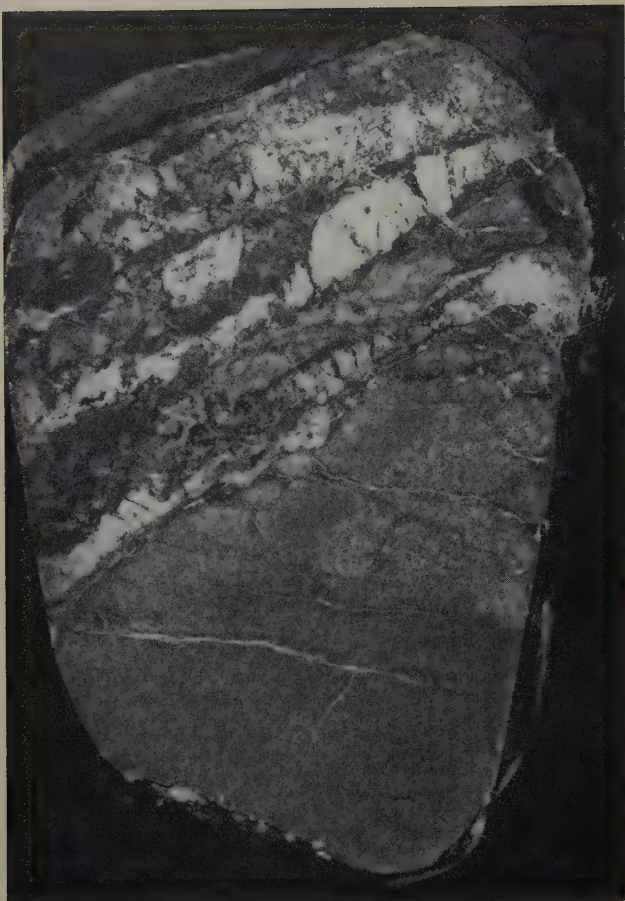


Fig. 16a (Left)—Red jasper cut by veinlets of pyrite mixed with and bordered by uraninite, 7th floor, 16 W. stope, 3100 level, area shown is 1.75x2.25 in. Fig. 16b (Above)—Radio-gram of pyrite-uraninite veinlets in Fig. 16a.

lected at the junction of a uraninite shoot and a flat fault. Red jasper, fine-grained pyrite, and uraninite form an earlier vein zone cut by a vein of coarsely crystalline siderite and later pyrite with traces of tetrahedrite. It is a familiar mode of occurrence of uraninite at the Sunshine mine. The coarse-grained pyrite, which is late, is light in color. With the black uraninite there is a vein of earlier pyrite; this is fine-grained and dark in color.

Uraninite occurs along with massive pyrite in vein segments 2 to 4 in. across, see Figs. 16a and 16b, on several levels. Pyrite-uraninite veinlets are encased in red jasper. Fine-grained pyrite (white) is bordered and separated in places by uraninite (black). Thin veinlets of later tetrahedrite cut both the uraninite area and the jasper. The higher concentration of uraninite and absence of siderite indicate uranium precipitation accompanied by silicification prior to the main carbonate vein formation, see Figs. 12a, 12b, 13, and 15.

Thurlow and Wright (p. 401 of ref. 1) have pointed to the progressive replacement of quartz by uraninite. Pyrite appears to have been similarly replaced. However, specimens may be found in which uraninite formed both before and after pyrite. Pyrite intimately associated with the uraninite, however, is either fine-grained or colloidal in dimensions in contrast to coarse or euhedral pyrite of later origin.

Early observations by Thurlow and Wright (p. 404 of ref. 1) indicate that uraninite is a late mineral in the Sunshine deposit and presumably not subject to zonal distribution. On this basis it is concluded that uraninite would not be expected to become more abundant with depth. The current study would differ with the conclusions of Thurlow and Wright with respect to the relative roles of tetrahedrite and uran-

inite. Only occasionally is uraninite found in siderite veins associated with tetrahedrite. In such occurrences the uraninite is isolated and, it is believed, represents fragments transported mechanically from earlier masses as occasional quartz or pyrite fragments are transported, Figs. 17a and 17b. While recrystallization may take place because of the action of new fluids, and microscopic textures simulating replacement may result, the prevailing mechanism is one of mechanical transport.

Thurlow and Wright (p. 401 of ref. 1) interpret a small micro-area of tetrahedrite (X750) as undergoing marginal replacement by uraninite. However, it appears that the replacement is not entirely marginal and it is suggested that the uraninite itself may be in part replaced.

The few places where later uraninite veinlets have been observed constitute a comparatively minor feature. The current study would place emphasis on an early origin for most of the uraninite. If uraninite is an early mineral, as suggested, zonal distribution of uranium in relation to silver might indicate greater concentration at a lower level.

Segmentation: The uraninite veins are ordinarily short segments, at most only a few feet long and often not more than a few inches in length. Even under the microscope, Fig. 18, colloidal aggregates of uraninite are segmented and parted by quartz.

Segmentation as shown in a stope is illustrated in Fig. 10. Northwesternly earlier uraninite segments are cut by siderite veins with an easterly normal strike. Further segmentation appears in Fig. 14. Localization accompanies segmentation as shown in Fig. 9. Although the veins may be observed to extend for several feet in the exposure, the chief ore-bearing lenses are 3 to 4 in. thick at the widest portion and taper to a fraction of an inch within a foot.

Lateral Variation: The groups of parallel veins and stringers which make up the vein zone vary from hanging wall to footwall. Solid siderite may occur on one side with strong uranium mineralization opposite, or various other combinations may be found. The veins shown in Fig. 10 vary from side to side. On the north side a siderite vein contains quartz



Fig. 17a—Streaks of uraninite in a siderite vein, 3100 level W.



Fig. 17b—Radiogram of Fig. 17a. Size 2.25x3.25 in.

segregations but no tetrahedrite. On the south side a parallel siderite vein is rich in tetrahedrite.

Siderite-Uraninite Occurrence

The siderite-uraninite relationship in the Sunshine mine is a matter of interest. Usually uraninite occurs in portions of veins where siderite is absent, but occasionally the two are found in close proximity. Whether the uraninite found with the siderite represents a later generation or whether it is merely reworked material that has been transported more or less mechanically is not always certain.

An illustration of the uraninite-siderite association is shown in Figs. 17a and 17b. The succession of vein opening and filling is complex. The lower half of the specimen consists of jasper formed with the fine quartz impregnated with finely crystalline arsenopyrite. The upper half of the specimen consists of successive bands of siderite; uraninite; pyrite; mixed siderite, pyrite, and uraninite; pyrite; and siderite containing pyrite and tetrahedrite.

Siderite is the prominent gangue mineral of the Coeur d'Alene silver veins. It may occur alone or impregnated with varying amounts of silver-bearing tetrahedrite. Massive veins may be observed with the hanging wall type of siderite, heavily mineralized with tetrahedrite. White quartz, apparently unmineralized, may occur in isolated patches. Evidently such veins do not bear uraninite.

Manganese is a prominent constituent of the carbonate, as shown by X-ray fluorescence curves. Small amounts of other elements may be present but do not yield peaks above background. Calcium and magnesium were determined for a considerable number of carbonate samples by Mitcham.²

Thermal curves, Fig. 19, of siderite from the Sunshine mine yield an endothermic peak between 500° and 550°C, followed by a variable exothermic peak, ranging from 550° to 620°C.

Among the siderite samples chosen for thermal analysis were: a sample from a stope closely associated with red jasper, pyrite, and tetrahedrite

(3250 level, 25-ft stope raise); a sample from a stope containing siderite and tetrahedrite (31 to 16 W, 3rd floor); a sample of siderite mixed with quartz, pyrite, and tetrahedrite (Sunshine vein, 3100 level at junction of Jewell crosscut and main drift); a sample of siderite closely associated with uraninite (2nd radioactive vein, 3000 level); a sample of siderite associated with pyrite, quartz, and uraninite (3400 to 25 stope, 30 ft below 3250 level); a fairly clean sample of siderite with stringers of white quartz (3250 level, 25 E stope).

The samples selected were as clean as possible, but in most cases small amounts of fine-grained pyrite were present. Several reported variations in thermal curves of siderite may be due to varying particle size and tightness of packing, according to Rowland and Jonas.¹⁰ The thermal curves for sid-

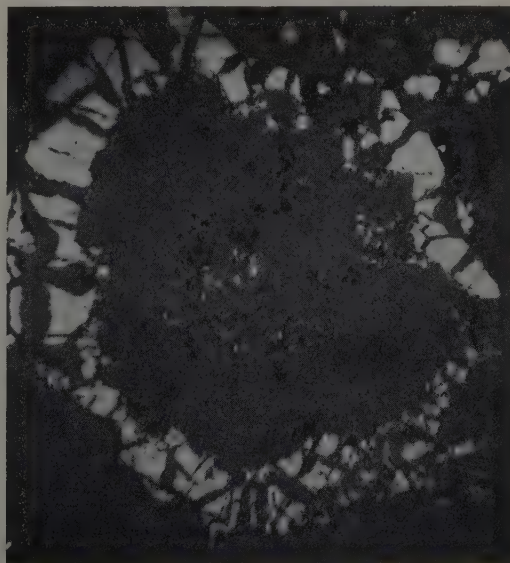


Fig. 18—Micrograph of a polished surface shows colloform uraninite forming a circular group of segmented fragments in a gangue of red jasper. X120.

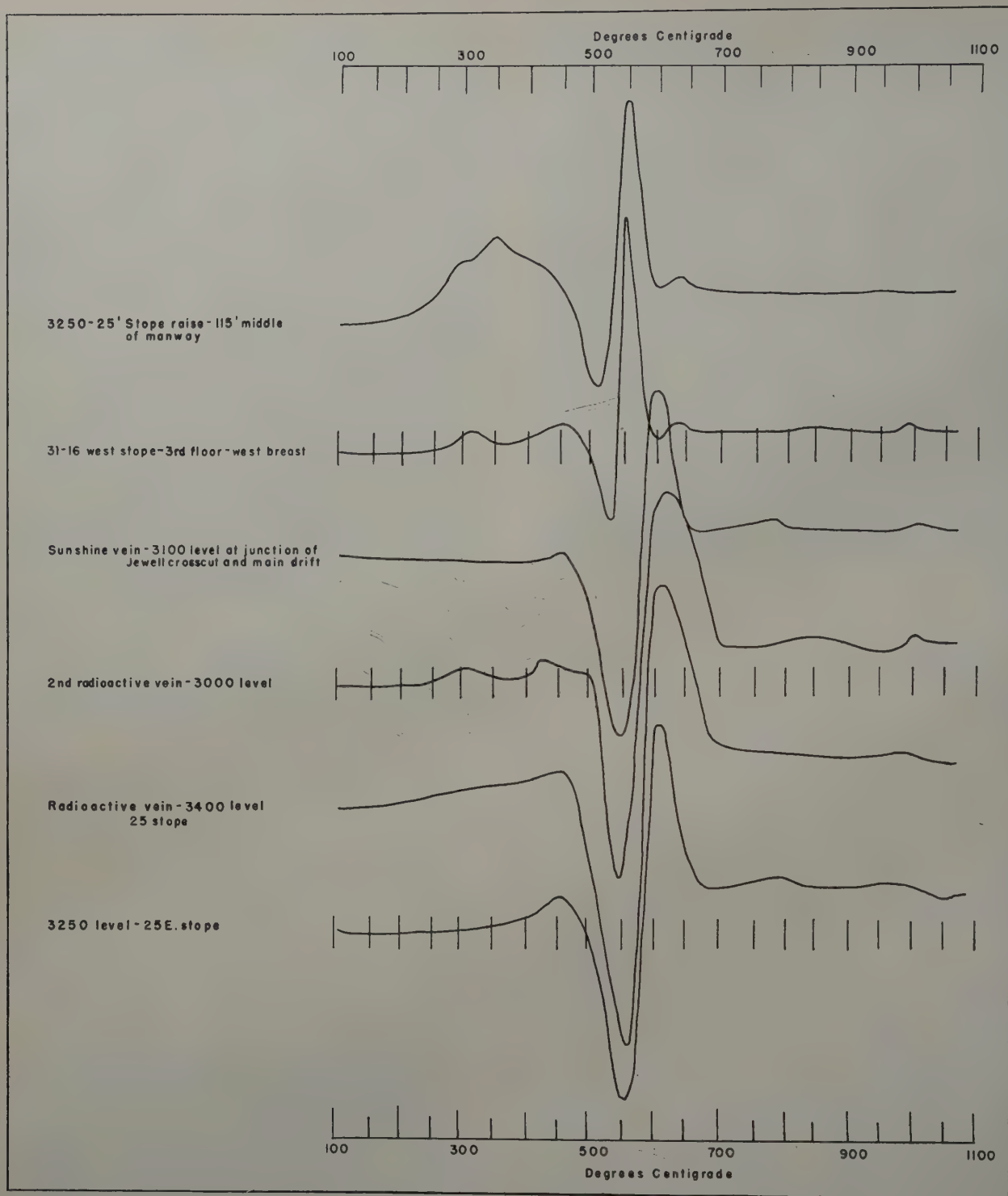


Fig. 19—Thermal curves of siderite from the Sunshine mine yield an endothermic peak between 500° and 550°C, followed by a variable exothermic peak, ranging from 550° to 620°C.

erite from the Sunshine mine are essentially the same whether the mineral is associated with uraninite or tetrahedrite or found in barren veins.

A specimen with a siderite vein bordered on each side by a thin streak of uraninite is shown in Figs. 20a and 20b. A color rim of red jasper immediately borders the uraninite. Apparently the fissure has been subjected to filling, reopening, and refilling; an earlier uraninite phase formed with a smoky border was later reopened and a later siderite phase was introduced.

Pyrite-uraninite veinlets bordered by uraninite halos and jasper are shown in Figs. 21a and 21b. These veinlets are truncated at an oblique angle by

a larger and later vein composed chiefly of siderite. A few scattered patches of tetrahedrite occur in the siderite vein but none were observable in the diagonal veinlets.

Many occurrences may be found in which it appears that siderite veins were formed at an early stage. Such veins are usually much thinner than siderite of later origin. Other criteria consist of more extensive fracturing, segmentation, and intersections with later siderite, as well as structures indicative of reopening and filling.

Fig. 22 represents two generations of siderite, both forming steep veins; a later finer-grained brown siderite forms veins a foot thick, whereas

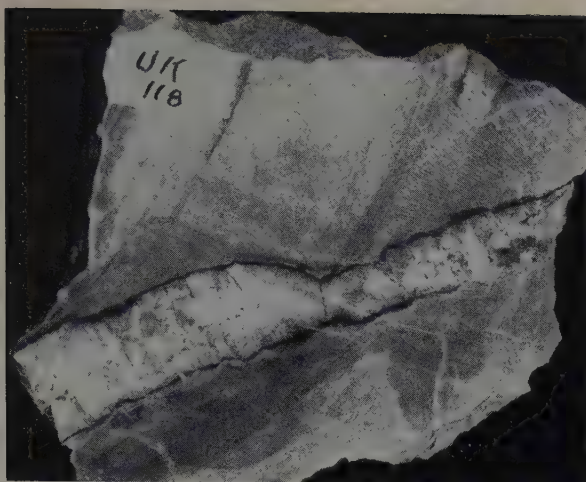


Fig. 20a—Siderite vein with uraninite margins, 25 E. stope 40 ft below level, area shown is 4.5x5.5 in. An earlier uraninite



vein apparently was reopened and siderite deposited in the center. Fig. 20b—Radiogram of Fig. 20a.

earlier coarser veins are thin. Quartz ladder veins lie in the thick veins in fractures due to shrinkage or tension. A single remnant stringer of coarse brown siderite is enclosed in the thicker hanging wall siderite vein. The following sequence may be inferred: 1—bedding shear and fractures filled with earlier siderite; 2—associated red quartz accompanying nearby uranium introduction; 3—later siderite veins; 4—white quartz filling cracks in later veins; late fractures.

One such earlier vein may be seen in the area between the Sunshine and Chester veins, Fig. 23. Here a vertical siderite vein has been terminated by the uraninite-bearing jasper, while siderite stringers, presumably filling tension cracks, appear to be later. The jasper-uraninite mineralization followed the vertical siderite and presumably preceded the siderite stringers. A nearby white quartz vein is isolated but is probably later.

In a number of specimens uraninite is concentrated along the margins of tetrahedrite-siderite veins or siderite veins as already mentioned, see Figs. 20a and 20b, 21a and 21b, and 23. In such specimens the original fracture carrying uraninite appears to have been reopened and filled with siderite. A late siderite vein may be seen along the 3100 W. drift, Fig. 24, some 3 to 6 in. thick, which penetrates a red jasper area containing disseminated uraninite.

The siderite vein itself contains a ladderlike pattern of quartz veins. Uraninite along the brecciated zone illustrated in Fig. 23 shows a later siderite cutting jasper and earlier uraninite. The various features of a uranium-bearing zone cut by non-radioactive siderite-tetrahedrite veins are illustrated in Fig. 25.

Variation in the tetrahedrite content of siderite veins is noteworthy. Frequently, as shown in Fig. 10, one vein of a group may contain all or nearly all of the tetrahedrite for a cluster of several veins which vary in size. Such tetrahedrite-bearing veins are ordinarily subsequent to the uranium mineralization and cut areas of red jasper. Tetrahedrite ore normally occurs in a filling in coarse siderite veins. Small patches of pink quartz are occasionally associated with such veins or even minor amounts of uraninite, but for the most part coarse tetrahedrite-siderite veins are free from uranium.

The minerals of the Coeur d'Alene have been listed by Mitcham.² Few sulphides in addition to pyrite and arsenopyrite are found in the uranian area. On the 3400-level crosscut to the Chester vein, see Fig. 23, surrounding small uraninite concentrations, is an irregular mass of red jasper containing disseminated chalcopryite, tetrahedrite, and uraninite for a width of 18 to 20 in. A specimen from the 3700 level, Jewell crosscut vein, shows multiple

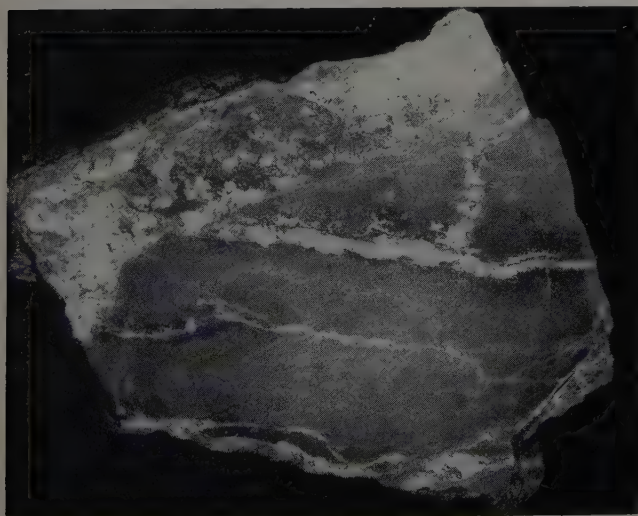


Fig. 21a—Siderite vein cutting early pyrite-uraninite veins.



Area shown is 4.5x5 in. Fig. 21b—Radiogram of Fig. 21a

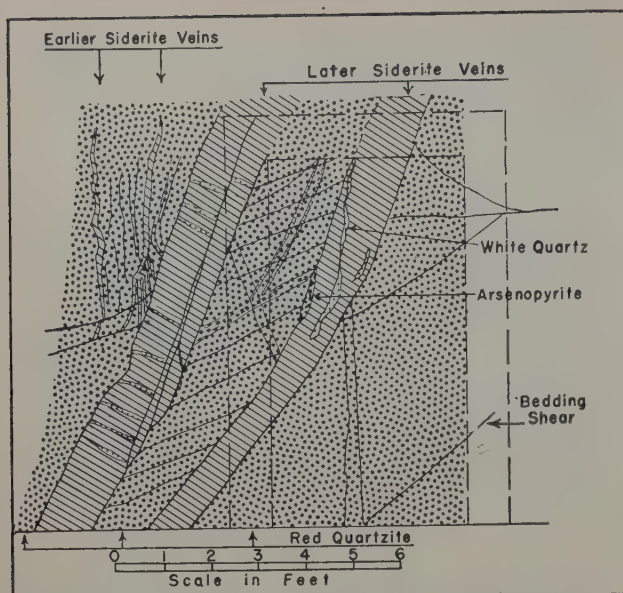


Fig. 22—Two generations of siderite veins, S. W. face, 16W. stope, 3100 level, 3rd floor.

stages of pyrite, quartz, siderite, and some stibnite. Another from the 3700 level shows a red quartz-siderite-pyrite vein containing galena, cutting black radioactive carbonate, and pyrite, Figs. 26a and 26b. This specimen is unusual because in this instance galena is found in a vein cutting a radioactive zone. As may be seen from the radiogram, the uraninite is intimately mixed with the carbonate and pyrite on each side of a galena-bearing vein, but the vein itself is not radioactive.

Unmineralized white quartz occurs in ribbon veins, Fig. 2b, where it is probably contemporaneous with uraninite-bearing quartz, Fig. 2a. It also occurs in cracks, see Figs. 22 and 24, within siderite veins. A later generation, Fig. 6, with minor galena, occurs as crosscutting veinlets. A white quartz veinlet about an inch thick and crenulated in outline with scattered small patches of galena may be observed in Fig. 11 cutting earlier ribbon vein structures at right angles.

An unusual white quartz vein containing galena and boulangerite (?) occurs along the uraninite-bearing zone on the 3700 level. The quartz vein is parallel to the main vein for the most part, but gradually cuts across the course of the uraninite vein. It may represent a change in mineralization with depth, since galena-bearing veins are unusual at higher levels in the Sunshine mine.

Thin white quartz veins about $\frac{1}{4}$ -in. thick may occur in parallel lines isolated from the main vein in the footwall. Such barren veins occasionally lie close to parallel veins containing siderite and tetrahedrite.

Indications of Uranium Mineralization

Certain significant characteristics accompanying the uranium deposition are indicative of mineralized zones, the most striking being the stain of red, brown, or pink in finely crystalline silica near uraninite. Radioactivity along fractures and in gouge is also indicative, but it may not lead directly to ore, since with available equipment radioactivity may be detected in concentrations considerably below present mining limits. Another feature is the association of arsenopyrite. Where arsenopyrite is abun-

dant, either high uranium or high silver or both are usually found. Presumably, arsenopyrite was one of the early sulphides and was precipitated in heavy concentrations within a comparatively narrow range.

Coloration in Crystalline Silica

The red alteration or the so-called jasper has not yet been adequately explained, but in the Sunshine mine several features appear to be established.

Repeated association of red with uraninite and the absence of coloration where uraninite is not found demonstrate that the jasper and uraninite are related. Also the color is always confined to a siliceous matrix, the silica being microcrystalline to colloidal. The jasper zone varies from a fraction of an inch to several inches in width and is a localized phenomenon, see Fig. 27.

The jasper has long been noted by the miners in the Sunshine mine, although uraninite itself was not recognized until recently. It has not been reported from the levels above 2900. While geiger counters were not available when the upper levels were mined, it is probable that the red halos, had they existed, would have been observed when the rock was freshly broken. The zones of red alteration indicated by exposures along drifts, in crosscuts and cores from drill holes, see Fig. 28, form a broader and more regular pattern than the corresponding uranium occurrences, Fig. 3. The pattern of the red zones suggests close folding in the St. Regis quartzite on the 3100 level west of the Jewell crosscut. Aside from an occurrence in the Jewell crosscut, 150 ft from the Sunshine vein, and a sporadic occurrence at the east end of the workings, Fig. 28, the red alteration lies on the south limb of the minor anticline as indicated in Fig. 3. A 45° drill hole driven southward at the west end of the 3100 drift encountered red alteration, at 36 to 59 ft, although a nearby horizontal hole driven in the same direction failed to encounter the alteration. The zone is believed to show a rake to the west under the second drill hole.

In general, depth of color increases with uranium concentration. Several dozen fragments selected to show the entire range of color were examined with a geiger counter. With few exceptions which could be explained by variations in size, the highest counts were given by the most highly colored lumps and

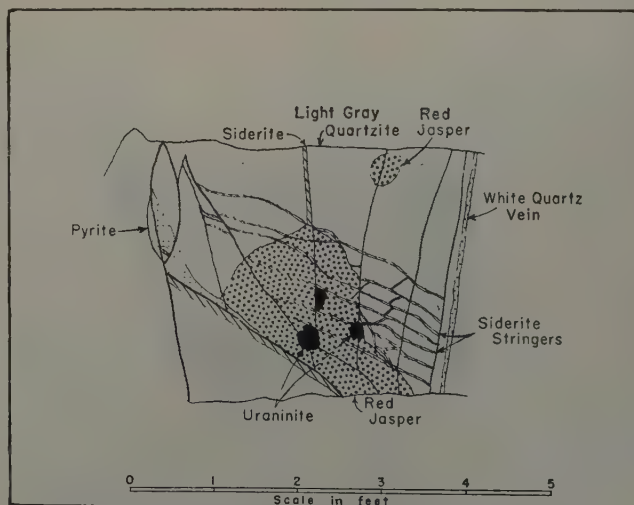


Fig. 23—Uraninite occurrence from a brecciated zone about midway between the Sunshine vein and the Chester vein. It represents both pre-siderite and post-siderite deposition.

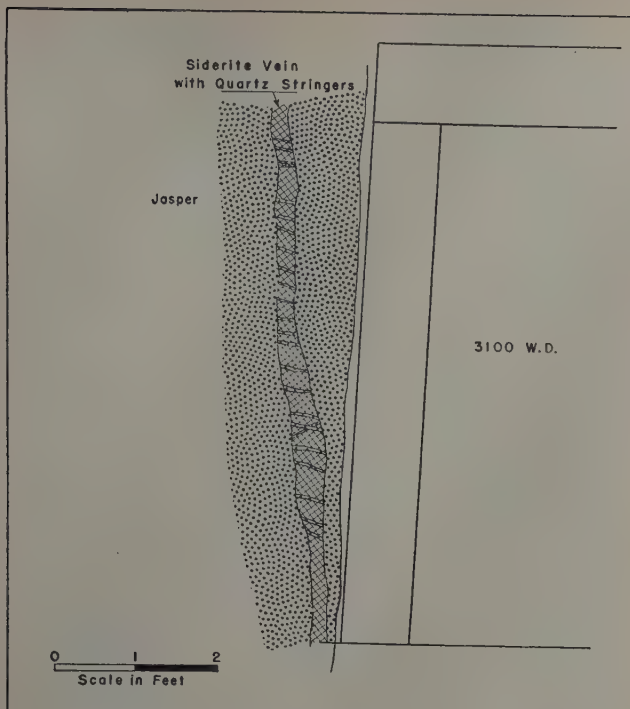


Fig. 24—Ladder quartz in a late siderite vein 3 to 6 in. thick, which penetrates a red jasper area containing uraninite.

counts decreased with a decrease in the intensity of color.

When the jasper is examined with the microscope it is found to contain disseminated grains of metallic minerals. Most abundant are disseminated grains of sulphides, chiefly pyrite, but scattered grains of uraninite are more plentiful in areas where the color is most intense. Other fine sulphide grains are arsenopyrite, Fig. 29, sphalerite, and chalcopyrite. Occasionally larger patches of tetrahedrite may be observed, but these appear to follow fractures and probably represent later introduction.

The pigment responsible for the color is fairly uniformly distributed and resembles the red of agates artificially colored by absorption of ferric hydroxide followed by roasting. A possible explanation would be that a small amount of ferric oxide formed contemporaneously with uraninite is absorbed in the finely crystalline or colloidal silica of the matrix.

Red alteration is characteristic of the borders of uraninite veins at Great Bear Lake, in the Beaver Lodge area and elsewhere in Canada. Dawson¹¹ has attributed the red color associated with uraninite at Ace Lake to fine hematite distributed through oligoclaseite. The fine-grained siliceous rocks of Canadian areas are often comparable in color to the Sunshine material. The alteration is also accompanied in many cases both by an increase in iron content and radioactivity. The mineral carrying the uranium is uraninite, or pitchblende, and the iron is ordinarily present as hematite. While this is a general feature for Canadian occurrences, exceptions have been noted.

Specimens collected on the 3100-level drift, at about 75,000 E, show pink quartz. A hanging wall siderite vein contains brecciated carbonate with streaks of pinkish quartz cutting the carbonate and apparently replacing it. The pink quartz carried small amounts of carbonate which may be of a different type. Veinlets containing the quartz are discontinuous and lenticular and contain little uraninite. Black fractures in the specimens are limonite-filled. White quartz is present in the siderite, but there is apparently no metallic mineralization associated with the white quartz of this type. It would seem that white quartz forms areas of little uranium or silver mineralization.

Occasionally smoky black halos of alteration occur along pyrite-uraninite veinlets, the halo fading into gray-brown jasper. Both the halo and the veinlets may be cut by siderite veins which contain isolated patches of tetrahedrite but no uraninite. Disseminated uraninite with less hematite appears to give the halos a smoky appearance.

Arsenopyrite Zones

Arsenopyrite frequently accompanies the uranium-bearing veins. It may be present as an earlier vein filling disseminated throughout the quartzite, Fig. 9, or in masses along vein borders. Although associated with siderite in some earlier phases, it is more commonly found with quartz. Where evidence of sequence is available, it is pre-uraninite. Mitcham,² in a study of the tetrahedrite veins of the Silver Belt, looks upon arsenopyrite as an indicator mineral in connection with silver orebodies. Evidence from sections examined in connection with uranium mineralization would point to a similar significance as an indicator for uranium deposition.

Fig. 25 — Uranium zone silver cut by silver zone, 3rd floor, 16 W. 3100 level. Earlier uraninite veinlets bordered by jasper and disseminated arsenopyrite in quartzite are cut by later nonradioactive siderite-tetrahedrite veins bearing silver.

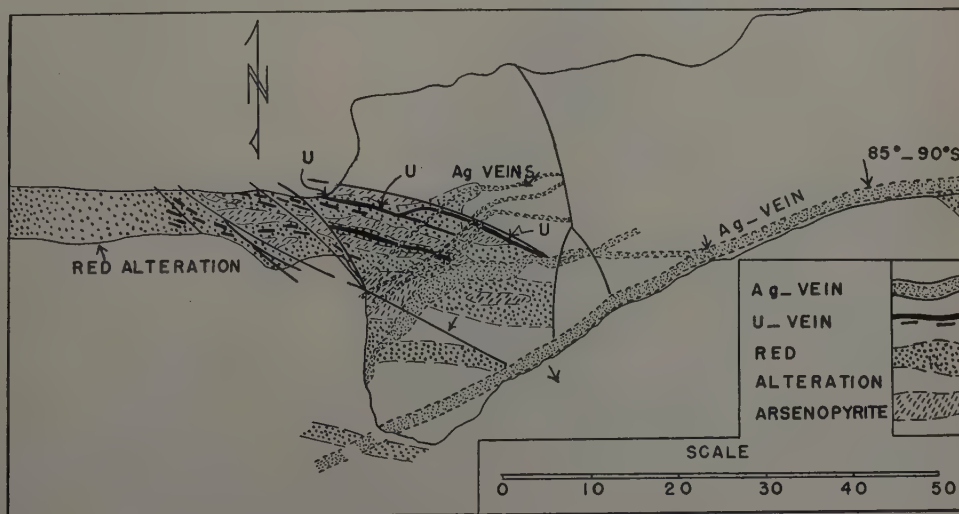
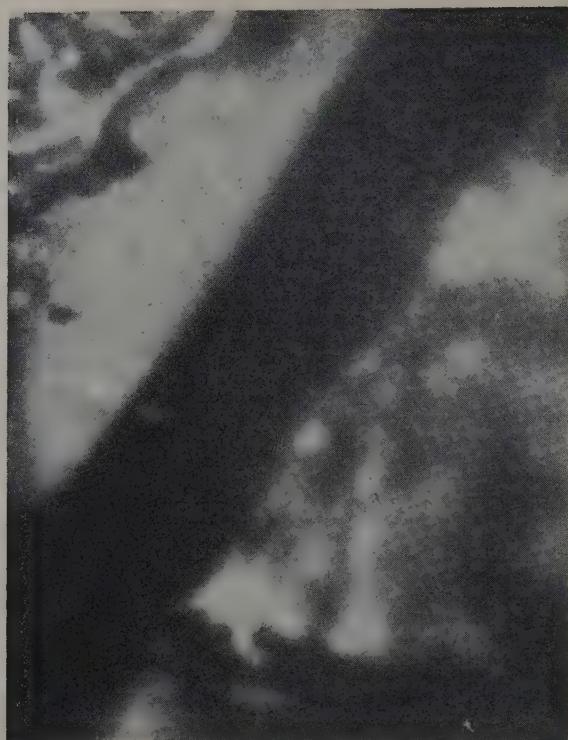




Fig. 26a—A red quartz ankerite vein containing galena cutting black radioactive carbonate, 3700 level about 4 ft East



of Jewell crosscut. Area shown is 3.5x8 in. Fig. 26b—Radio-gram of Fig. 26a.

Where arsenopyrite is abundant, solutions responsible for early stages of vein mineral precipitation contain a high concentration of metallic elements. Thus in the vicinity of such places it might be expected that there would be massive pyritic veins and the heaviest concentrations of uraninite. This may be one occurrence of uraninite, but examination of several specimens would indicate that uraninite is distributed widely and many times occurs in places devoid of arsenopyrite.

Arsenopyrite may occur in minute crystals both in the quartzite of the vein walls and in the veins themselves. Wall rock fragments, Fig. 16a and 16b, isolated and partly replaced by pyrite and siderite masses, are in places impregnated by arsenopyrite. At one place in a zone of fracture cleavage, Fig. 9, arsenopyrite is distributed in the footwall locally forming an arsenopyrite schist. Some of the lenses of uraninite are well developed at this point. Arsenopyrite is usually unaltered, but occasionally, in old workings, a secondary yellow mineral resembling orpiment has developed. A prominent arsenopyrite-uraninite association is shown in Fig. 8. Here substantial veinlets of uraninite were observed. Further, both the footwall of the vein area and a streak along the hanging wall for a distance of 25 ft are impregnated with arsenopyrite.

A significant feature in mineralization observed in places along barren ribbon veins which might give a clue to the proximity of nearby uranium concentration is dark gray quartzite impregnated at times with arsenopyrite.

Disseminated arsenopyrite, Fig. 25, appears related to the uranium zones at times, but not to the silver zones. The trend along uraninite stringers is cut by silver veins. Arsenopyrite areas also appear to be encircled by red alteration.

Arsenopyrite vein stringers are shown in Fig. 8. Here both the footwall of the vein area and a streak

along the hanging wall for a distance of 25 ft are impregnated with arsenopyrite. The exposure indicates vein shearing and fracturing, followed by arsenopyrite vein filling. Then comes uranium-bearing red jasper, siderite veins, and post vein fractures.

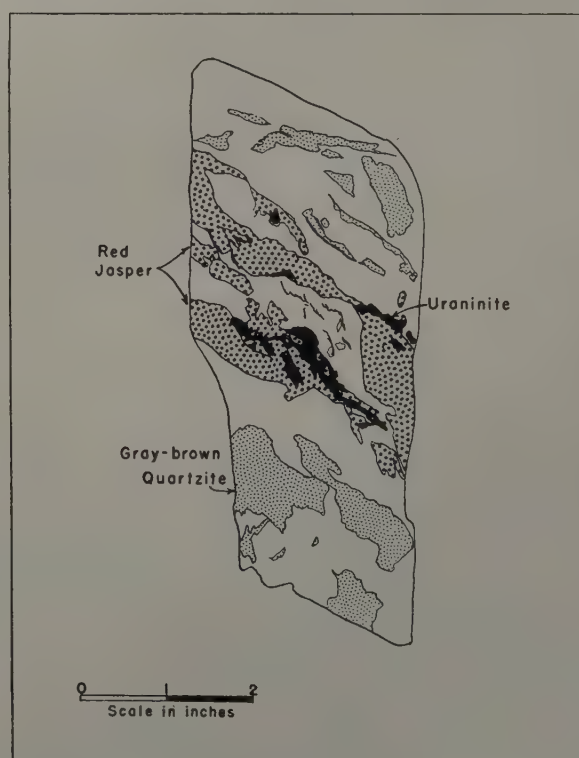


Fig. 27—Specimen diagram of jasper-uraninite association in quartzite, uraninite (black), red jasper (dots), and grayish-brown jasper (stippled). Streamers of pyrite are visible in the uraninite.

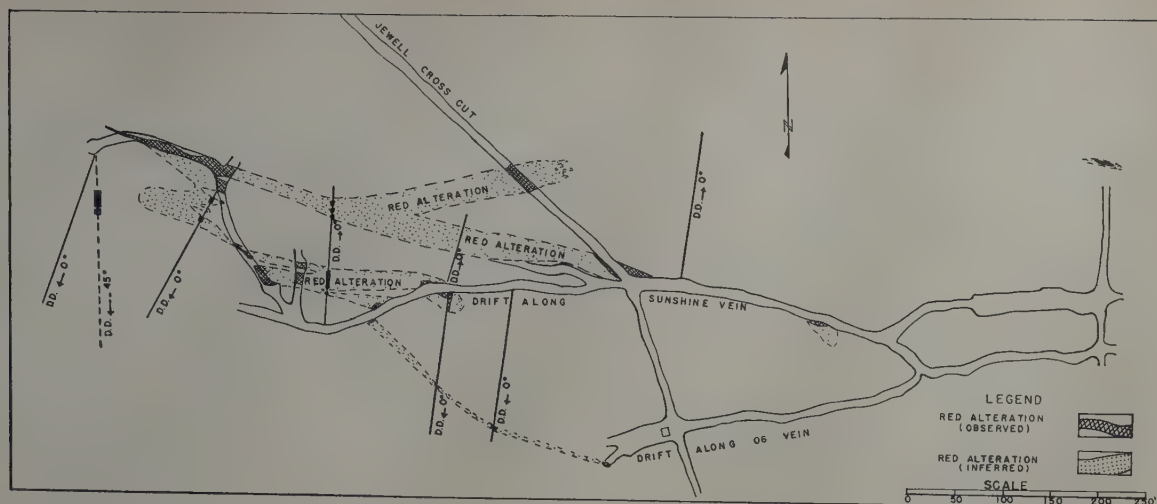


Fig. 28—Red alteration along 3100 level. Observed zones of red alteration jasper are shown in relationship to inferred zones based upon exposures in the workings and drill cores.

As previously noted, the presence of uranium was first detected through a reconnaissance survey with a geiger counter. Since then workings have been surveyed with geiger counters, and where possible drill holes have been examined with radioactivity detection devices. These studies have outlined the radioactive area.

Radioactivity

A radiometric diagram of the 3100 level, Fig. 4, shows seven positive anomalies with gamma counts exceeding 500 per sec; two occur along the Jewell crosscut and five along the west drift. An abrupt change from high to low radioactivity across shear zones appears in two places. At one, west of No. 4 shaft, a low angle bedding plane fault terminates the uranium-bearing zone. Red alteration and higher counts occur above the fault and to the west, while below the fault and to the east the count is low and red alteration is absent. At another, N 40° to 50°W, shear 80 ft west of the Sunshine-Jewell intersection locally limits westward extension of red alteration in the 3250 to 25 E stope 25 to 50 ft below the point.

Contour readings are given as low as 30, but the normal background is 50 to 60 counts per sec; hence it is not likely that such low contours are significant. The radiometric diagram is based on a scintillometer survey by R. F. Robinson, M. Reyner, and J. Coulson. Readings were taken at 5-ft intervals in areas of high values, but at 25 ft elsewhere.

While the radiometric diagram is inadequate to justify quantitative application, it reveals a pattern which corresponds in general to distribution and red alteration of uranium mineralization. It also emphasizes the localized nature of the radioactivity.

Uraninite: Analytical Data

Through the cooperation of the Division of Raw Materials, AEC, both chemical analyses and lead isotope analyses of uraninite from the Sunshine mine have been secured. The material analyzed was collected on the west face of No. 16 stope, 3100 level, west of Jewell crosscut. X-ray data on the uraninite yield a lattice constant of 5.444 Å. Table I gives the indexed powder data, in Å* units, for the uraninite

* X-ray measurements are given in terms of Angstrom units based upon Siegbahn's wave lengths.

from the Sunshine mine.

Table I. Analysis of Uraninite, in Å Units, From Sunshine Mine

I	2θ (Cu)	d(Å)	(hkl)
100	28.68	3.116	(111)
50	33.24	2.692	(002)
80	47.51	1.912	(022)
80	56.28	1.631	(113)
20	58.94	1.565	(222)
10	69.30	1.354	(004)
30	76.32	1.246	(133)
30	78.77	1.41	(024)
30	88.04	1.108	(224)
30	94.80	1.046	(333), (115)
10	106.62	0.961	(044)
40	113.88	0.919	(135)
20	116.14	0.908	(006), (244)
20	127.11	0.860	(026)
20	136.27	0.830	(335)
20	139.63	0.821	(226)

The sample analyzed was hand-picked but in view of the impurities shown in the analysis must be assumed to have contained pyrite, siderite, tetrahedrite, quartz, and sericite. The determinations re-

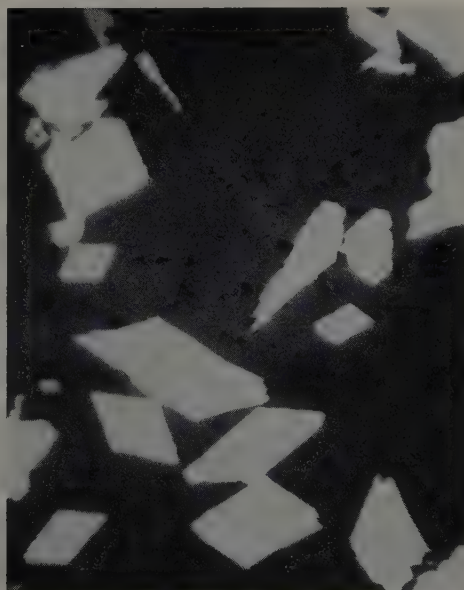


Fig. 29—Micrograph of rhombic sections of arsenopyrite in quartz. Vein pyrite is transected and replaced by numerous quartz veinlets, which in turn are replaced by arsenopyrite.

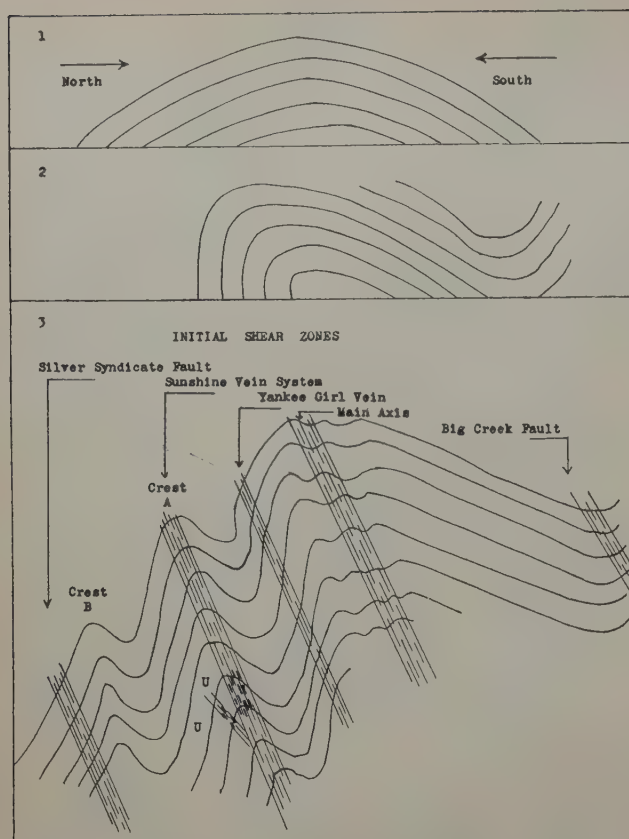


Fig. 30—Initial folding and fracturing, sectional diagram.

ported by Clara Gale Goldbeck of the New Brunswick Laboratory, AEC, are given in Table II.

Table II. Chemical Analysis, Sunshine Mine

Item	Pct
U ₃ O ₈	26.9
PbO	4.0
SiO ₂	18.2
Al ₂ O ₃	4.4
Fe	13.1
CaO	.8
Mn	1.0
Cu	7.7
Ag	1.0
Sb	2.6
H ₂ O	1.1
CO ₂	6.8
S	9.0

Isotope analyses of the uraninite, secured through the cooperation of Roger Hibbs, Carbide and Carbon Chemicals Co., Oak Ridge, and reported by Kerr and Kulp,⁸ are shown in Table III.

Sequence of Emplacement

The emplacement of the ores of the Sunshine mine has consisted of a complex process of successive fracturing and filling. Sections and specimens, however, provide data from which the major sequence may be interpreted, see Figs. 30 to 32.

Deformation of the Beltian sediments provided the major structural features along which ore emplacement of both the silver and uraninite ores has taken place. As a result of major compression, Fig. 30, steeply inclined fractures, zones of fracture cleavage, local folds, breccia, cross faults, and bedding plane slips were developed, all of which par-

ticipated in one way or another in providing channels for the penetration and localization of the ore-forming fluids.

The steeply inclined fractures have provided loci for the precipitation of silver ores, but the uraninite has shown less limitation to the major vein fractures. Above the 3250 level at least uraninite has favored shear zones related in a number of instances to fracture cleavage. On the 3700 level, however, the occurrence is largely restricted to a fissure vein.

Table III. Lead Isotopes, Sunshine Uraninite

208	17.84 ± 0.07	0.460 ± 0.013	50.67 ± 0.05
207	12.42 ± 0.05	7.32 ± 0.02	20.69 ± 0.03
206	69.20 ± 0.03	92.17 ± 0.01	27.27 ± 0.05
204	0.540 ± 0.010	0.044 ± 0.001	1.37 ± 0.02

The first stage in uranium mineralization appears to have been the precipitation of uraninite along with pyrite under more or less colloidal conditions. In places arsenopyrite and pyrite penetrated the quartzite of the wall rock at about this stage. At the peak of emplacement quartzite close to areas of

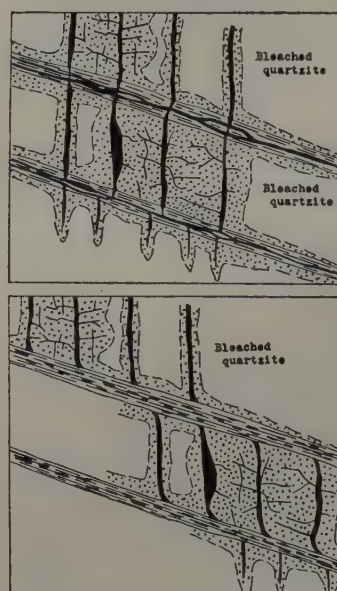


Fig. 31—Uraninite segmentation along flat faults.

uraninite precipitation was recrystallized and permeated by hematite and minute specks of uraninite resulting in a red halo. Little siderite appears to have developed where this process was most intense.

Following the main emplacement, the uraninite-bearing areas were subjected to faulting and in the main the early epoch of pyrite-quartz-uraninite mineralization appears to have been succeeded by the gradual development of strong carbonate veins accompanied by argentian tetrahedrite.

No clear lines of demarcation appear to separate the two epochs. Some siderite and a little tetrahedrite may be found in veinlets associated with early uraninite. Likewise fragments of uraninite and some streaks of radioactive material may be found at times in tetrahedrite-siderite veins. In general, however, the weight of evidence provided by sections and specimens points to an earlier uranium and a later silver epoch.

Deformation features following the second metallic epoch are less clearly defined but are shown by

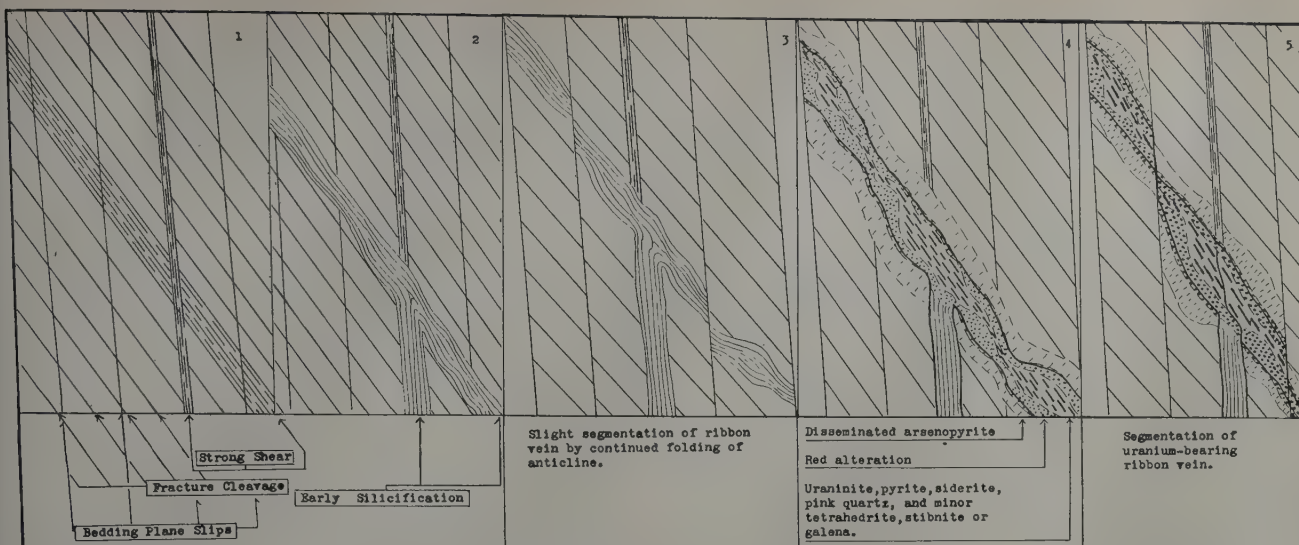


Fig. 32—Sections 1 and 2 show fracture cleavage and bedding plane slips with shearing and silicification as developed on the north limb of the overturned anticline on the 3700 level. Sections 3 to 5 illustrate the development of a ribbon vein carrying uraninite by replacement along a silicified shear zone with subsequent segmentation.

displacements along siderite-tetrahedrite veins and crosscutting structures. The same may be said for the post galena-quartz deformation.

The principal epoch of deformation after the start of mineralization appears to have followed the uraninite-pyrite stage and accompanied the introduction of siderite and tetrahedrite. While many small cross faults were formed by the inter-epoch movements and frequent reopening along vein courses resulted, the movements were seldom large, often being measured in inches and only occasionally more than a few feet.

In summary, the main sequence of events accompanying mineralization in the uraninite area may be interpreted as follows:

1—Regional deformation of Beltian sediments: major anticlinal structure; faulting along north limb; bedding slips, fracture cleavage; shear zones. 2—Early uraninite-pyrite-quartz emplacement accompanying arsenopyrite in wall rock: red halos in areas of greatest concentration; colloidal effects; some early siderite. 3—Intermineralization deformation: vein reopening and parallel fracturing; bedding plane, and flat faults; segmentation of uraninite veins. 4—Main tetrahedrite-siderite epoch: major silver veins formed; some solution and reprecipitation of uraninite. 5—Post silver deformation: parallel fracturing on a minor scale; limited cross faulting. 6—Quartz-galena stage: barren or low grade siderite veins; quartz-galena veins; white quartz veins. 7—Post mineral deformation.

Acknowledgments

Appreciation is expressed for assistance from many quarters. Mr. R. D. Leisk, General Manager, Sunshine Mining Co. and his staff associates have both aided the study and made mine data available for publication. Mr. Ernest Thurlow of the Division of Raw Materials, AEC, has aided in discussing field observations. Mrs. Clara Gale Goldbeck, Chief, Microchemical Branch, New Brunswick Laboratory, AEC and Dr. Roger Hibbs, Chief, Mass Assay Laboratory, Carbide and Carbon Chemicals Co., Oak Ridge, have provided analytical data. Professor J. L.

Kulp, Columbia University, has provided age computations. Mrs. Roselyn J. Steinhart and Mr. Edward Lyden, Research Assistants, Columbia University, have aided the laboratory investigation.

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Selection of Conveyors for Handling Hot Bulk Materials

by J. Walter Snavely

PRESENT-DAY processing in many industries, calcining, sintering, briquetting, beneficiation and nodulizing, increasingly calls for the handling of large volumes of hot bulk materials. Various types of conveyors have been employed. This discussion will cover the factors governing their selection.

For temperature ranges up to 400°F, or approximately 200°C, a wide range of conveyors is available. Special constructions of rubber conveyor belts, steel conveyor belts, vibrating and shaker conveyors, apron conveyors, and drag chain conveyors, all are used successfully. As temperatures go well above 400°F, however, choice of conveyors is narrowly limited. This paper will consider the problem of handling bulk materials only where the temperatures exceed 400°F.

The arbitrary selection of 400°F as a dividing point undoubtedly can be challenged, as special conveyor belting constructions are available which are suitable for temperatures in excess of 400°F. However, when the relatively short life of such belts and the cost of their replacement, with the attendant down time, are balanced against the reliability and long service life of the properly designed steel constructed units to be discussed, there is little question in any operator's mind that the special belts are more expensive to use.

Because the conveyors under study are for the handling of bulk materials, inevitably including a high proportion of fines, obviously wire mesh belts cannot be included for consideration. Even though this type of conveyor is widely used at high temperatures, i.e., for carrying glassware through alehr, it is unsuited for the conveying of bulk materials, and therefore will be excluded from further discussion in this paper.

Preliminary to the study of the conveyor itself is the determination as to whether the material is to be cooled while it is being handled, or whether the processing requires retention of all heat and the maintenance of a given temperature range.

In the majority of cases cooling is incidental to or part of the handling process, when the handling, for example, follows completion of sintering, roasting, calcining, refining, or some other process. To meet such operating conditions successfully, the conveying medium used must have: 1—a construction capable of withstanding maximum initial temperatures of the material being handled, 2—a construction providing efficient heat transfer for cooling. 3—a construction providing dependable operation and long life with minimum service requirements, and 4—a construction providing controlled and efficient conveying.

Under the usual conditions of cooling during the handling, the construction selected to withstand the initial maximum temperatures does not necessarily

involve using alloys, as excellent results can be achieved with normal carbon steels and cast irons, when they are properly applied and proportioned.

The earliest and simplest type of conveyor for handling very hot materials is the cast steel drag chain conveyor, still widely used for handling hot cement clinker, as illustrated by Figs. 1 and 2. Because of the rugged and generous proportions of the chain link design, low carbon steels are entirely suitable for the links. The pins, however, must be alloy steel. The simple, rugged construction of this type of conveyor makes it readily capable of withstanding high initial temperatures, even though the chain is operating buried in the material.

The drag-chain type of conveyor has advantages and limitations. Although the efficiency of the heat transfer is relatively poor, the life of the conveyor is reasonably long, and because of its crude simplicity it does not require much servicing. However, as a conveyor, it is limited in capacity, and largely limited to horizontal runs. Furthermore, because of the crude design, heavy weight, and the chain operating at the temperature of the material, greatly reducing permissible operating chain pulls, this type of conveyor is limited to relatively short centers.

Another type of conveyor that has been used for very hot materials is the cast pan conveyor. Because of its very generous proportions the cast pan, which is made of either cast iron or malleable iron, can withstand initial maximum temperatures. It also provides efficient heat transfer for cooling. Further, it is an efficient conveyor construction, which can be used for inclines. Because the chain employs rolling friction instead of sliding friction, and is not in the maximum temperature zone, much longer centers are possible.

It is this type of conveyor that is frequently used in the casting of various metal pigs, pig iron, and aluminum; it is obvious, therefore, that very high initial temperatures are being handled. With this kind of conveyor the return run is frequently sprayed with water to accelerate heat transfer. The build-up of residual heat in the very heavy cast pans is thus overcome.

The outboard roller steel pan conveyor is an improved pan conveyor which provides high rates of heat transfer and substitutes formed steel pans for the heavy cast pans. It is a very efficient conveying medium. The details of this particular construction are clearly shown in Fig. 3. An early application of this type of conveyor is shown in Fig. 4. In this case the conveyor units are handling roasted phosphate rock at average temperatures of 1000° to 1500°F, and frequent maximum temperatures as high as 1900°F. Several widths are used. The capacity of the unit at a speed of 50 fpm is approximately 30 tph per inch of width at peak loadings, average capacity being about 1/3 of peak loading.

The assembled conveyor is shown in Fig. 5, with views of both the top and the underside to show all the construction details. In particular, the following general design principles were carried out in this construction:

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Discussion on this paper, TP 3529B, may be sent (2 copies) to AIME before July 31, 1953. Manuscript, April 14, 1952. New York Meeting, February 1952.

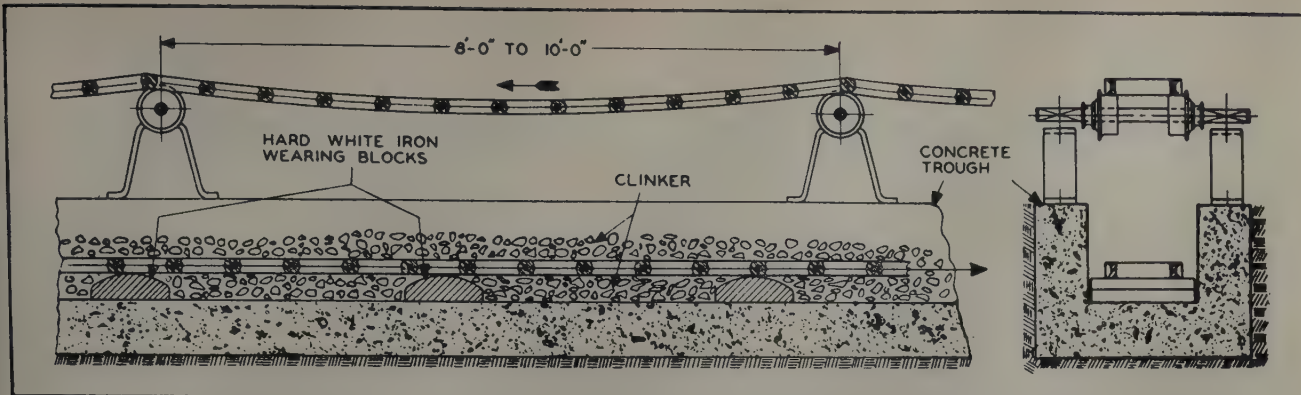


Fig. 1—Economical construction of a ground-level trough for hot or cold cement clinker.

1—The capacity of the conveyor, that is, the size of the pan and the speed, together with the weights of the steels used, was designed so that the mass of the carrying pan is considerably greater than the mass of the live load of hot material. This ratio should be approximately 2 or 3 to 1 for peak load conditions. The maximum initial temperatures are thus easily handled and any localization of intense heat is no problem.

2—The shape of the pans provides maximum surface area for efficient cooling.

3—The operating speed is low, approximately 50 fpm, permitting complete cooling on the return run and also greatly reducing the rate of wear.

4—The chain and carrying rollers are insulated from the heat zone.

5—The wear is largely confined to heavy, carrying, outboard rollers, on sleeve-type bushings, both

made of heat-treated white iron, extremely hard and long-wearing.

6—The function of the chain is solely to pull the load, and the through-rods connecting the two strands of chain, visible in the view of the bottom side of the conveyor, further prolong chain life by providing exceptional rigidity.

7—The unique outboard roller construction permits easy servicing without the need of any disassembly of the chain or conveyor. Individual servicing or replacement of rollers, bushings, or even pans, can be done without disturbing the chain or the remainder of the conveyor.

8—The lubrication requirement is simple. Graphite, applied in a suitable vehicle such as kerosene, which quickly evaporates, is all that is required. Anti-friction bearings are not practical in the outboard rollers because the seals are difficult to install

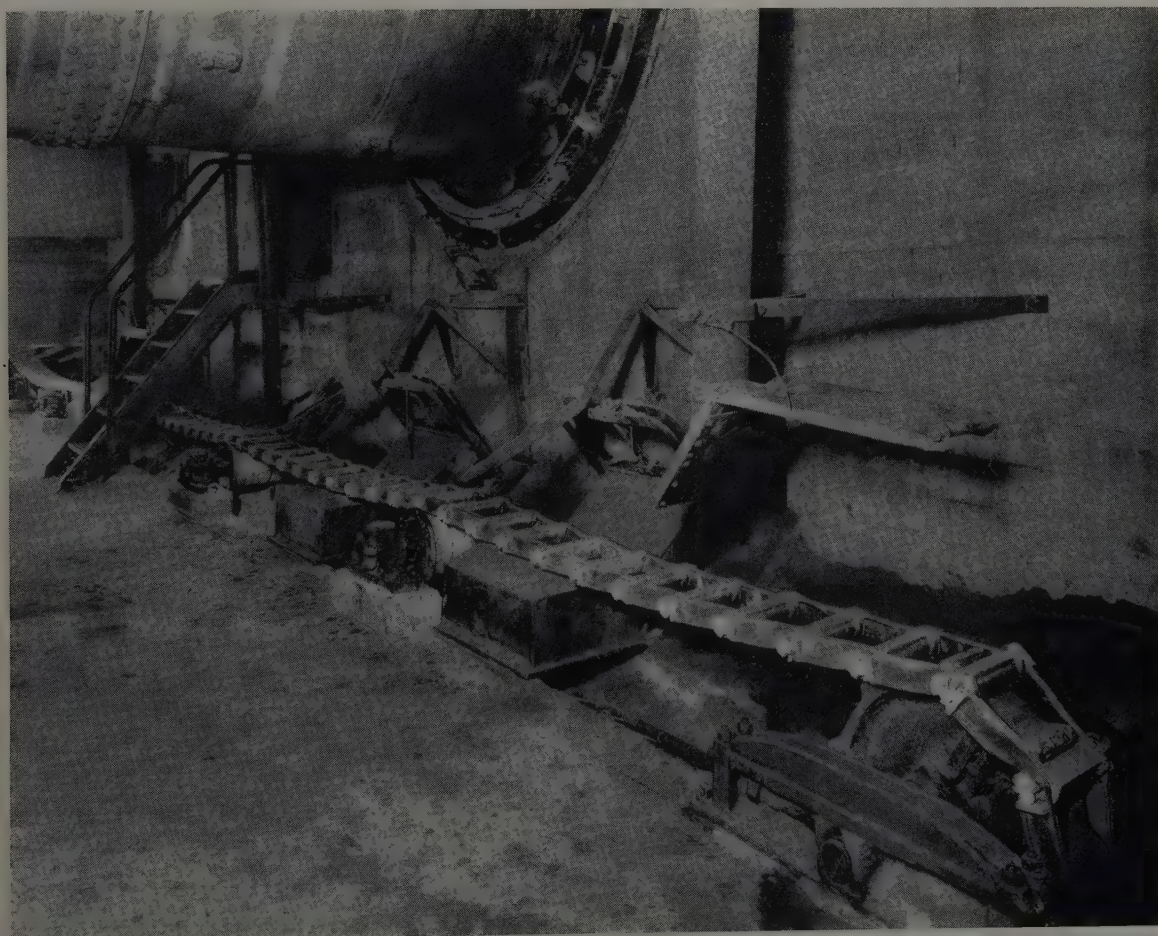


Fig. 2—Hot clinker and clinker dust are transported by the Rex drag conveyor illustrated. No. 123 cast steel drag chain belt is used.

and maintain, and, further, it is difficult to retain conventional lubricants in the anti-friction bearings under the heat conditions prevailing.

9—The relatively low power requirement, resulting from the use of the large diameter carrying rollers, and the shape of the pans, suitable for handling up steel inclines, make this construction an efficient conveyor medium.

A study of the foregoing principles reveals why other types of conveyors have not proved successful for the handling of materials at elevated temperatures. With steel conveyor belts, either carbon or stainless steel, the ratio of the mass of hot material to the mass of the belt is high, intensifying the effect of the localization of heat. Moreover, because the belt cannot be loaded full width out to the edges, the temperature of the belt is not uniform across its width, setting up severe expansion problems within the belt, causing buckling at the center and subsequent cracking as it goes around the terminal pulleys.

Vibrating conveyors have also proved unsatisfactory because of their poor heat transfer. The shape of the trough provides only nominal radiation area, and there is no return run for further cooling. As a result, the heat retention causes warping and distortion, which prevents the unit from conveying.

Leakproof Bucket Elevators

Where the vertical elevating of hot bulk materials is required by plant space limitations, bucket elevators can be used equally effectively if the same general design considerations are followed. Fig. 8 illustrates a special construction of slow speed, continuous bucket elevator that is in successful use handling calcined bauxite, highly abrasive, at initial temperatures of 1200°F.

The detailed cross section of the construction shows the entirely leak-proof design of the bucket, with overlapping construction. This feature

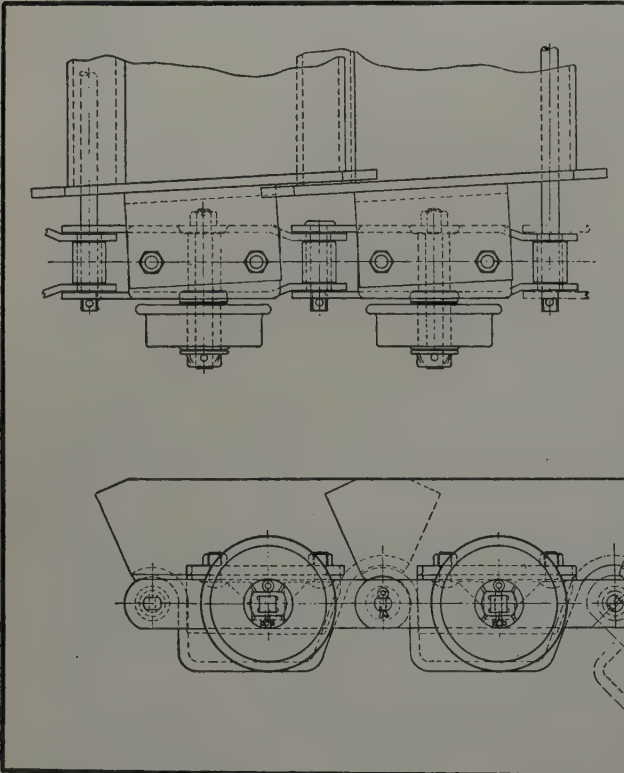


Fig. 3—Style C apron conveyor. Upper left, plan view. Lower left, side view. Lower right, end view.

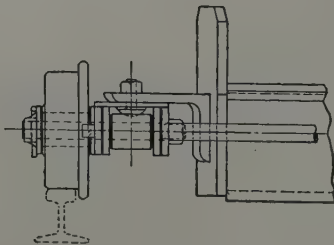
Table 1. Chains to Use at High Temperatures

Type of Chain	Maximum Temperature, °F	Heat Colors	Remarks
A.S.A. standard steel roller	350		
Extra clearance roller chains	500		"Nitroalloy" bushings for best wearing qualities Non-Corrosive as well as heat-resistant
Replacement series roller chains			
Stainless steel (18-8) block or roller chains	800		
	900	Faint red visible in twilight	
Monel metal block, stud or leaf chains	1000	Red visible in daylight	Nitrided pins for best wearing qualities
Cast steel chains (rollerless)	1050	Blood red	Low carbon steel links, alloy steel pins
Stainless steel (18-8) stud or leaf chains	1600	Bright red	
	1650	Salmon	Scaling temper, 18-8 stainless
Inconel nickel-chromium alloy (80-14) stud or leaf chains	2100		A wrought material extremely scale-resistant
Cast nichrome V 20-80	2100	Light yellow to white	Detachable or pintle type
Cast chrome A 20-80	2100		Detachable or pintle type

also makes possible very slow operating speeds. Normal construction of continuous-type bucket elevators does not provide for overlapping buckets, and small gaps open between buckets going around the terminals. A bucket speed of approximately 125 fpm is therefore required to throw material across these gaps to prevent spillage and back-legging. With the complete overlapping construction of Fig. 6, both on the bucket sides and on the bucket back and front, speeds of 60 fpm or slower are practical.

This bucket elevator design carries out the following general principles, which are parallel to the considerations for the pan conveyor, previously described, for the elevating of hot materials:

1—The capacity of the elevator, that is, the size of the bucket and speed, and the weight of the steels used provide a mass of the bucket greater than the mass of the live load being carried. This should be approximately a minimum of 3 or 4 to 1 for peak loading conditions. Maximum initial temperatures



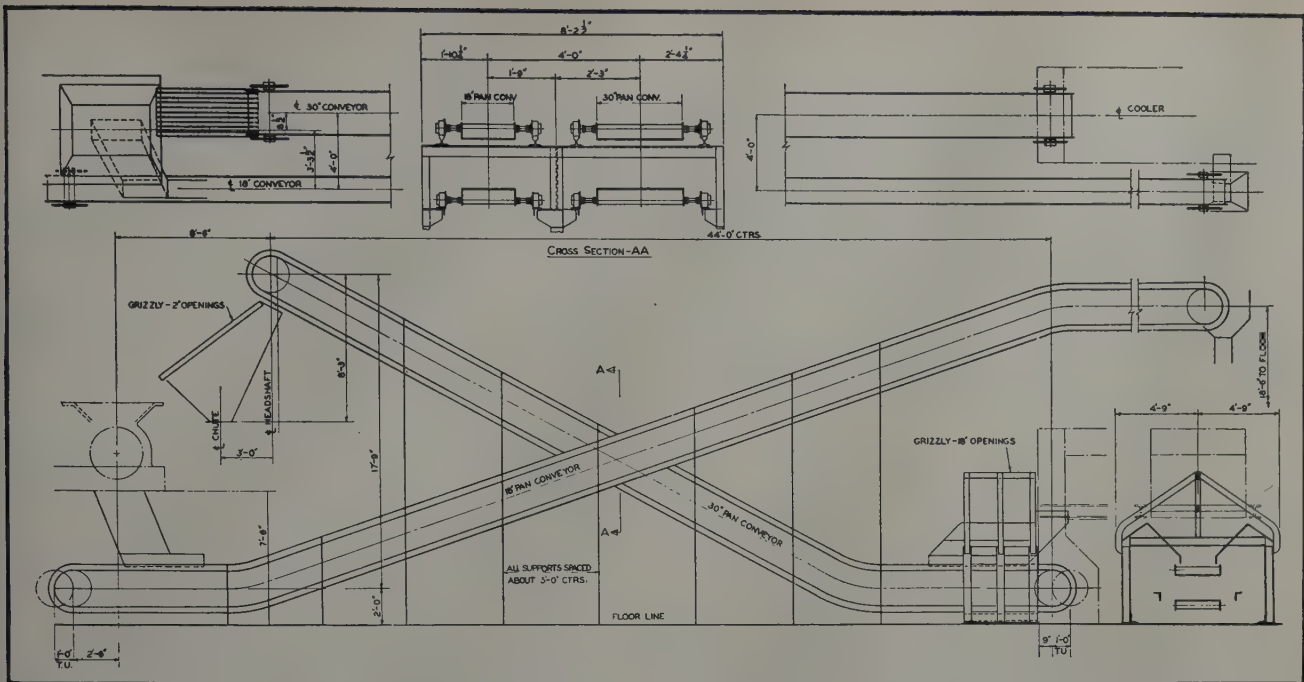


Fig. 4—Phosphate rock pan conveyors.

are easily handled without the necessity of special provisions.

2—The design and shape of the buckets, with the extended sides, bottom and back, provide maximum surface area for efficient cooling.

3—The operating speed is low, approximately 50 fpm, permitting efficient cooling on the return run and greatly reducing wear.

4—The chain is at least partially insulated from the high-heat zone.

5—The venting of the elevator casing helps to dissipate the heat.

6—The unique bucket design, which is entirely leakproof, keeps the chain completely clean, providing for longer wear and less servicing.

7—The design is a very efficient conveying medium, and in fact can be used as an accurate volumetric feeder.

A study of the foregoing shows why the attempted use of the en masse type of conveyor has been unsuccessful for the elevating of hot materials. With the en masse type of conveyor, the chain is completely buried in the material and therefore remains at maximum temperature. The surface area is only nominal, and the ratio of the mass of the hot material to the conveyor is much too high for efficient cooling. The cooling on the return run is also poor.

Further additional general considerations of both the constructions described are in order. It is obvious that normal conveyor capacity ratings and ratios of live to dead loads are discarded. Providing efficient heat transfer upsets usual relationships.

A higher ratio of mass of carrying medium to load is desirable for the bucket elevator than for the apron conveyor. This is true because the closer confinement of the elevator casing makes it harder to

Table II. Temperature Factors for Maximum Allowable Design Stresses

		TYPE OF CHAIN					
Temperature, °F		A. S. A. Standard Steel Roller	Replacement Series Roller	Rolled Monel Metal	Cast Steel Chains	Cast High Manganese Steel	Wrought Stainless 18-8
MODERATE TEMPERATURES	Room	70	100 pct	100 pct	100 pct	100 pct	100 pct
	200	100	100	95	100	89	86
	300	100	100	93	100	84	79
	350	100	100	93	100	81	76.5
	400		100	93	100	80	76.5
	500		100	93	100	76	76.5
	600		100	85	100	74	76.5
	700			70.5	100	74	73
	800			60	100	74	71.5
	850			54	92	74	70
	900			50.5	86	74	67.5
HIGH TEMPERATURES	950			48.5	76	74	63.5
	1000			46.5	64	74	61.5
	1050				56	74	
	1100						56
	1200						48
	1300						37.7
	1400						25.7
	1500						18.6
	1600						15.8
	1700						
	1800						
	1900						
	2000						

Above data has been computed on the basis of stress causing creep rate of 1 pct elongation in 10,000 hr.

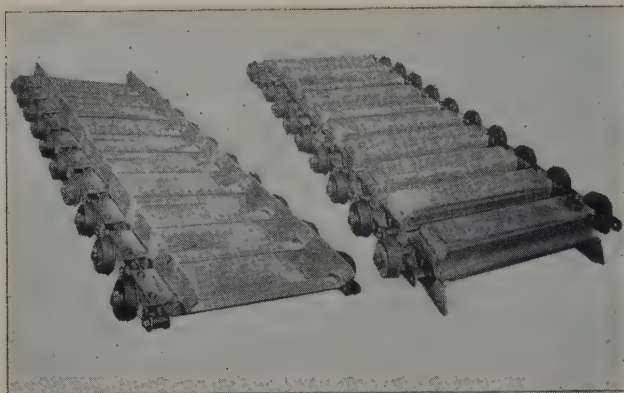


Fig. 5—Views of the assembled conveyor, top and underside, show the construction details.

remove the heat and because the chain is not so well protected from the maximum heat zone.

The slow operating speed increases the life of the working parts far more than would normally be expected. Rather than a straight line function, with the increase in life directly proportional to the decrease in speed, the relationship is a geometric one. This is comparable to increase in life in relation to the difference of the squares of the speeds. Slowing down the operation of the units therefore pays handsome dividends in service life.

In the design of wearing parts, conservative unit live-bearing pressures should be used, probably only 75 pct of normal ratings. While the temperatures of the working parts are not in the critical ranges, nevertheless the temperatures are above normal, effective lubrication is largely non-existent, and a high ratio of dead load to live load exists.

The selection of terminal equipment must like-

wise be tailored to the operating conditions. Such provisions as expansion joints for enclosures, gravity takeup arrangements, and bronze-bushed pillow block equipment, rather than anti-friction bearing or babbitted, are obvious necessities.

Material at Sustained Maximum Temperatures

Some processes demand sustained operation at the high operating temperatures. Where those conditions must be met, then the selection of the alloy materials must withstand the full effect of the temperatures over prolonged periods of time. Briefly, the problem is one of providing for creep, combating oxidation, and combating loss of strength as it affects working load capacities.

Various manganese, nickel, and chromium alloys can be used for operation at elevated temperatures. Based upon extensive study and experience on the part of the Baldwin-Duckworth Division of Chain Belt Co., Tables I and II present, in convenient form, the basis for the proper selection of alloys to be used at various temperature ranges.

Special Alloy Continuous Bucket Elevators

Outstanding examples of successful construction for sustained operation at high temperatures are the bucket elevators, used for handling the hot catalyst in the Thermoform Catalytic Cracking Refining Process, which were developed by Jeffrey Manufacturing Co.

Fig. 7 shows the flowsheet of the refining process. These bucket elevators must operate continuously 24 hr a day, 7 days a week, for a minimum of 9000 to 10,000 operating hr before any reconditioning, all the while handling highly abrasive material at 1000°F. The elevator casings must be gas-tight to insure operation in a completely inert atmosphere. Moreover, a number of these bucket elevators are

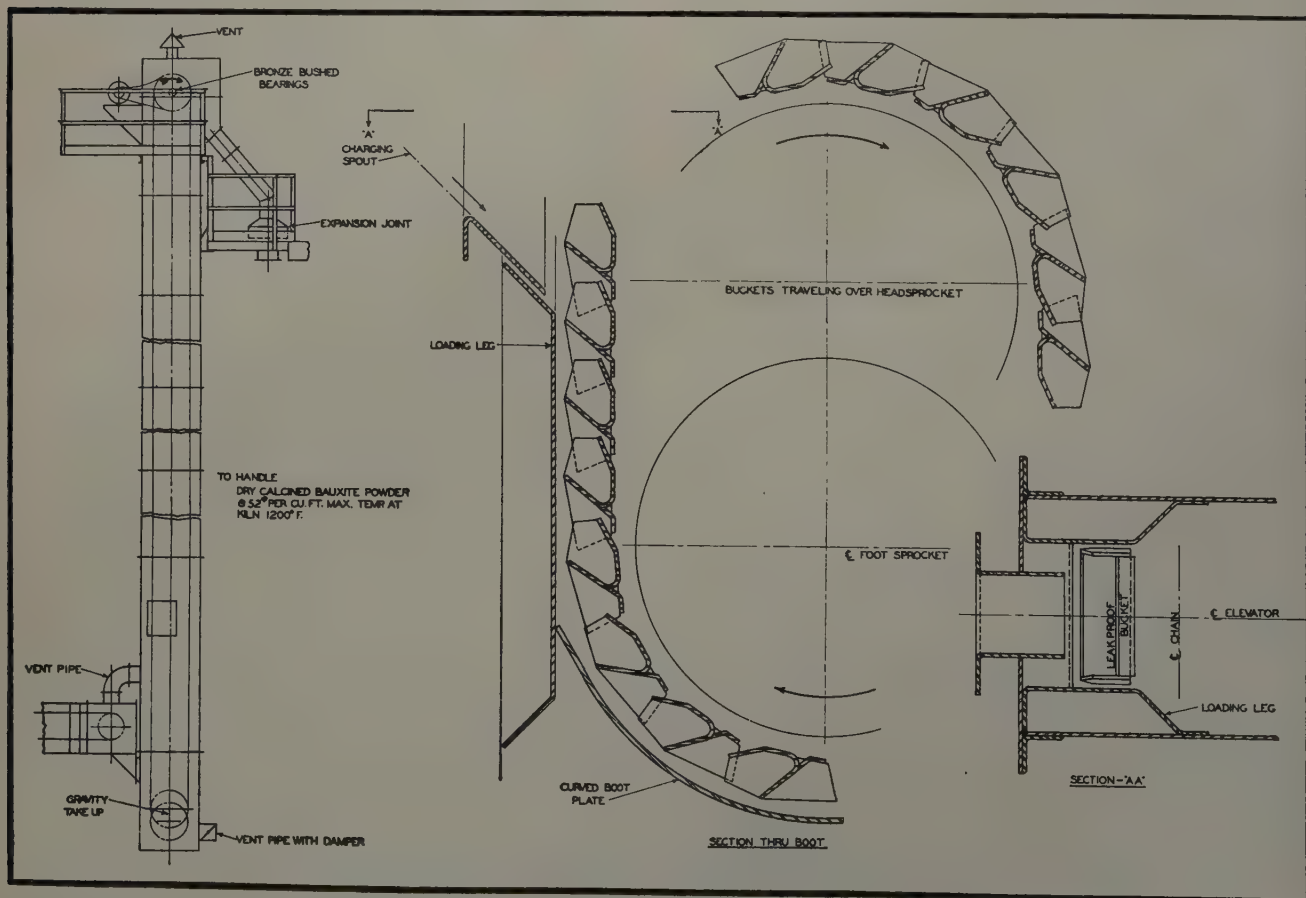


Fig. 6—Design of the leakproof bucket elevator.

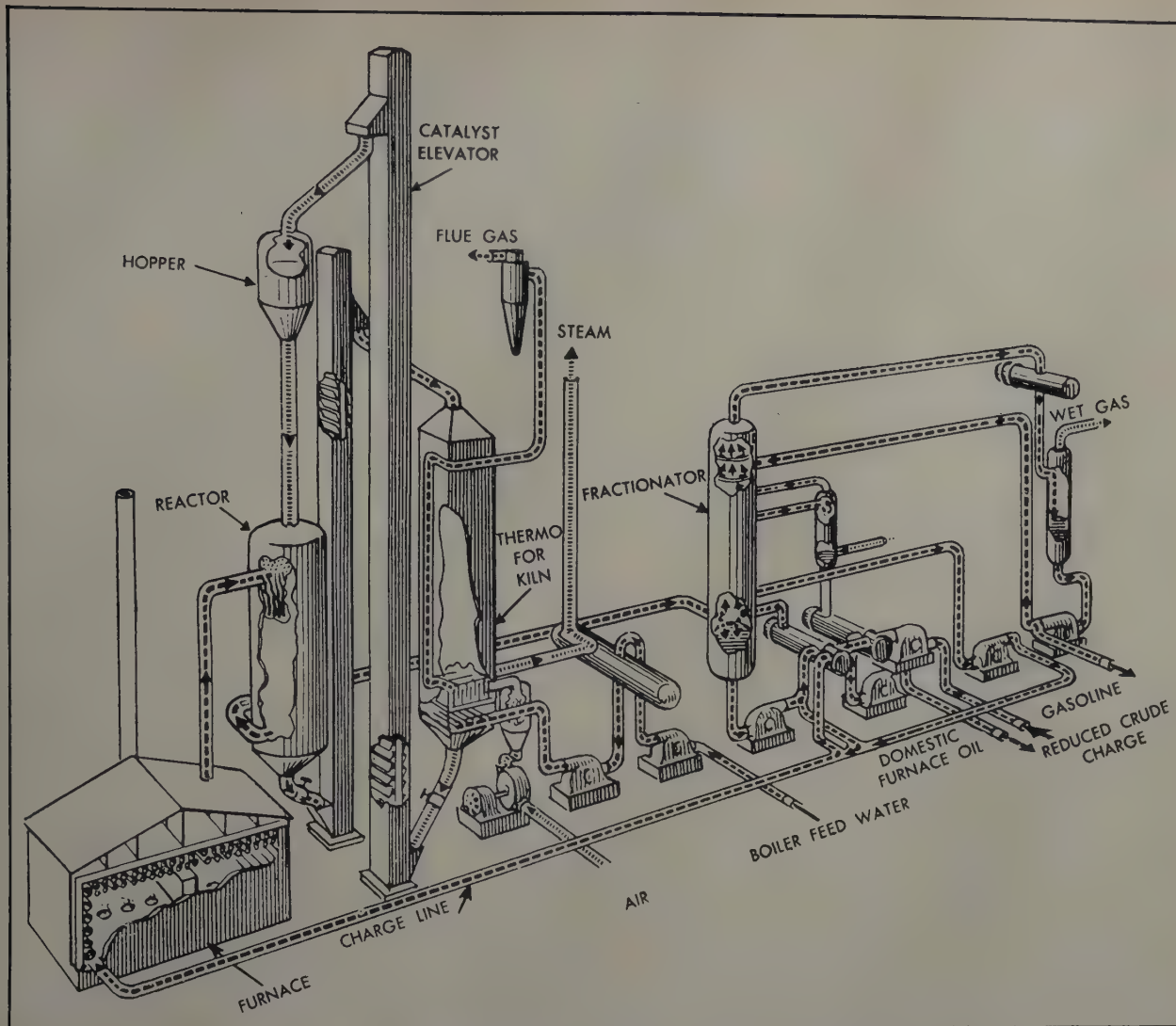


Fig. 7—General process flow diagram for TCC plant on liquid charge.

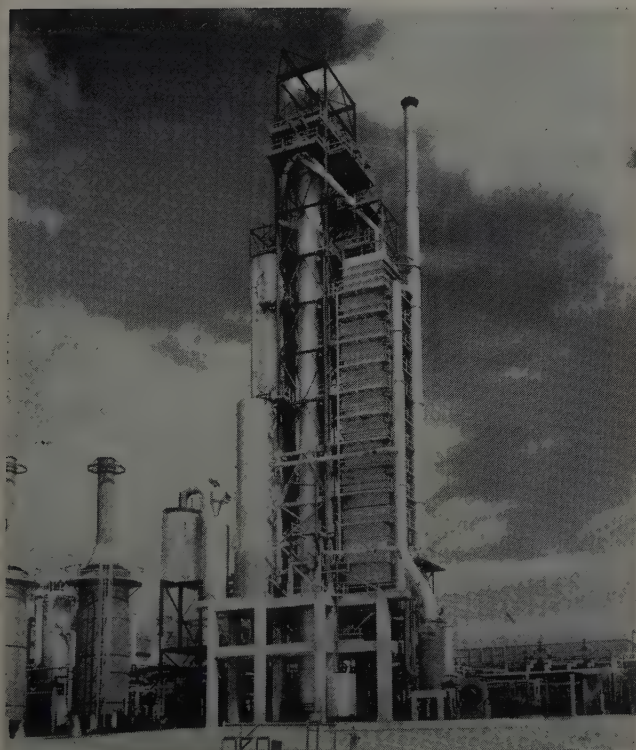


Fig. 8—View of the entire cracking unit. The bucket elevator is located within the cylindrical casing.

in operation handling 100 to 200 tph of 50 lb per cu ft material with centers of 200 ft, in itself a real design problem. Super units will soon go into operation handling 500 tph with a 300-ft lift.

The special chain was designed for proper strength, hardness, and creep rate. One detail of construction illustrates what can be done. The bushings are made of special heat-treated Ni-Hard castings, ground to size, and shrunk-fitted into the chain sidebars. At room temperature the hardness of these bushings is 600 Brinell and at operating temperatures 477 Brinell. The service life of the special chain has approached as much as 18,000 hr before reconditioning has become necessary.

The elevator casing is cylindrical, and as seen in Fig. 8, which shows the entire cracking unit, the bucket elevator is located within the structural steel work and would scarcely be recognized as a bucket elevator were that fact not previously known.

Acknowledgments

Acknowledgment is hereby given to the Engineering Department of the Baldwin-Duckworth Division at Chain Belt Co. for Tables I and II, to A. W. Lemmon, Consultant Sales Engineer of the Jeffery Manufacturing Co., Columbus, Ohio, for the information given on the T. C. C. bucket elevators, and to D. L. Schott, Chain Belt Co., Milwaukee, Wis., for criticism of the heat transfer phases presented.

Underground Radio Communication

In Lake Superior District Mines

by E. W. Felegy

THE need for improved mine communication to increase efficiency and to insure greater safety has long been recognized. General and unrestricted communication between all points underground, and between the surface and all points underground, is probably the least advanced phase of the mining industry.

An ideal system of mine communication must require no fixed wire installations. The equipment must be small, lightweight, and readily portable, and the power requirements low. A system that can be used not only under normal circumstances but also in an emergency, when the continuity of wires, tracks, and pipelines may be disrupted, must function independently of any aid furnished by standard installations.

Radio communication offers possibilities of meeting all the requirements necessary for an ideal communication system in underground mines. Transmission of signals must be achieved through one or both of two mediums, through the air in mine openings or through the strata. The results or lack of results obtained by early investigators showed conclusively that radio communication by space transmission cannot be accomplished effectively beyond line-of-sight distances in underground passageways. A radio system underground therefore must depend solely upon transmission through soil and strata.

The application of radio to underground mine communication was investigated by many individuals and agencies at different times in the last several decades, but little success was achieved before World War II.^{2-6, 8, 9, 11-16, 18, 19, 21} The results of experiments during the war, and further knowledge gained in experiments with vastly improved communication methods and equipment after the war,^{1, 7, 10, 17, 20, 22} provided the background for additional research in radio communication in underground mines.

During 1950 to 1952 the University of Minnesota sponsored an investigation to determine the possibility of developing a system of radio communication universally applicable in underground metal mines in the Lake Superior district. The possibility of using radio equipment to determine the imminence of rock bursts in deep copper mines in that district also was investigated. The investigation supplemented previous and concurrent emergency mine communication studies of the U. S. Bureau of Mines. Testing equipment and laboratory facilities maintained by the Bureau of Mines at Duluth, Minnesota, were used in the research program, which was conducted

as a mining engineering graduate research problem by the present writer under the direction of T. L. Joseph and E. P. Pfeider.

The radio units used in the research program were designed and built to specification solely to conduct tests of radio communication in mines. Two identical units were used in all tests.

Each unit contained a transmitter section, a receiver section, and a power-supply section, mounted on a single chassis. The entire unit was enclosed in a single 10x12x18-in. metal case provided with a leather-strap handle for carrying purposes. The front of the case was hinged to open upward and provide easy access to the single control panel on which all controls were mounted. Storage batteries supplied the operating power for all tests. Standard 6-v automobile batteries were utilized to provide adequate capacity to conduct tests for a full day without exhausting the battery.

A frequency range from 30 to 200 kc was covered in eight pre-fixed steps on each unit. The carrier frequencies were crystal-controlled and amplitude-modulated. The receiver employed an essentially standard superheterodyne circuit and was sufficiently sensitive to detect signal strengths of 5 micro v. A heterodyne circuit was employed in the transmitter to obtain the low-carrier frequencies used in the units. Power output of the transmitter, usually less than 2 w, rarely exceeded 3 w in any test.

Tests were conducted in mines on the Vermillion iron range in Minnesota, the Gogebic iron range in Wisconsin, the Menominee and Marquette iron ranges in Michigan, and a copper mine in the upper Michigan peninsula. All tests were conducted when the mines were operating normally, and usual mining, maintenance, and transportation activities were in progress, so that any interference caused by normal production activities could be evaluated during the tests. Tests were made between different points underground in each mine, and between underground and surface points at some mines. Test readings obtained at any one mine were calibrated in the laboratory before another series of tests were begun at the next mine. The transmitter and receiver were separated by one or more levels in each test, and generally there was no other means of communication between test points.

Two 100-ft lengths of rubber-covered wire were used for antenna wires on each unit in both transmission and reception. The ends of the wires were connected to ground points in one of several methods, depending upon physical conditions at each test site. The wires were clipped to metal rods about 200 ft apart in the back, side, or bottom of the mine opening where the character of the rock permitted driving rods. Both wires were clipped to points about

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Discussion on this paper, TP 3545A, may be sent (2 copies) to AIME before July 31, 1953. Manuscript, Sept. 2, 1952. Los Angeles Meeting, February 1953.

Table I. Summary of Test Results at Montreal Mine

Distance in Feet	Type of Strata	Wire Aid	Signal Strength	Noise	Remarks
270	Ore	Yes	High	High	
375	Ore	Yes	High	High	
260	Greenstone	No	High	Negligible	
504	Greenstone	No	High	Negligible	
525	Greenstone	No	High	Negligible	
1085	Greenstone	No	High	Negligible	
1135	Greenstone	No	Medium	Medium	
Over 1500	Greenstone, quartz slate	No	None	Excessively high	Failure to communicate possibly due to noise
1330	Greenstone, iron formation	No	None	Excessively high	Failure to communicate probably due to noise
1045	Greenstone, iron formation with faults and dikes	No	None	Excessively high	Failure to communicate probably due to noise
970	Greenstone, quartz slate, and soil	No	Medium	High	Local broadcast station interfered on surface; noise low underground

200 ft apart on the track or on pipelines in other tests, or one wire was clipped to the track or pipeline and one to a ground rod. Antenna wires of a radio unit on the surface for a test at a mine always were connected to metal rods about 200 ft apart in the ground.

To eliminate transmission of signals by wire circuits one or both radio units were set up, wherever possible, at tests sites without wire installations. It was also possible in a few instances to utilize sites having no pipelines and track. Despite the precautions observed in selecting sites, however, some individual results showed that wire circuits not in proximity to the radio units did aid in transmitting signals.

Frequency-response curves, obtained by plotting received signal strengths against carrier frequencies for each test, indicated whether communication was achieved through the strata or by means of wire circuits. Eve suggested the validity of such curves^{5, 6} after making tests in the Mount Royal tunnel in Canada and the Mammoth Cave in Kentucky. His conclusions were reaffirmed, most recently by Felegy and others,^{1, 7} after tests conducted in various mining regions in the United States.

A frequency-response curve shows maximum signal strengths received at the lowest frequency tested, and constantly decreasing signal strengths as the frequency increases when communication is achieved through the strata, see Curve a, Fig. 1. The general trend of a frequency-response curve is reversed, so that received signal strengths increase as the frequency increases where communication is achieved by wire aid; one or more peaks may occur in the curve between the lowest and highest frequencies, Curve b, Fig. 1. The difference in frequency-response curves usually is easily distinguishable. When the curves obtained in individual tests in this investigation indicated that communication was not achieved

solely through the strata, those tests were not considered in determining the feasibility of radio communication in underground mines.

Tests at each mine were concluded when the maximum distance through which communication was possible was determined at that mine, or when no other suitable test sites were available, or, as at one mine, when it was determined that communication through the strata was not possible.

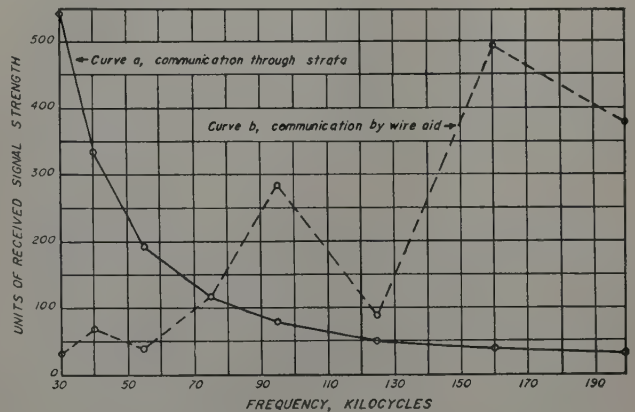


Fig. 1—Typical frequency response curves.

Fig. 2 is a diagrammatic section showing sites and relative positions of radio units in tests conducted at the Montreal mine, Montreal Mining Co., on the Gogebic iron range in Wisconsin. Table I is a summary of test results obtained at that mine.

Satisfactory two-way voice communication was achieved through a maximum distance of 1135 ft between underground points in the Montreal mine. Noise interference on the radio units was intense where the units were set up in or near operating sections of the mine. Satisfactory grounding of elec-

Table II. Summary of Test Results at Pioneer Mine

Distance in Feet	Type of Strata	Wire Aid	Signal Strength	Noise	Remarks
180	Greenstone	Possibly	High	Low	
340	Greenstone	No	High	Low	
775	Greenstone, iron formation	No	Medium	Medium	
1285	Greenstone, iron formation	Possibly	Low	High	
1000	Ore, iron formation, area faulted	No	None	High	Failure to communicate probably due to faults
1165	Caved jasper and soil	No	High	Low	
1330	Caved and solid jasper, and soil	No	Medium	Low	
1496	Caved and solid jasper, and soil	No	Low	Low	Maximum limit; insufficient signal strength to increase distance

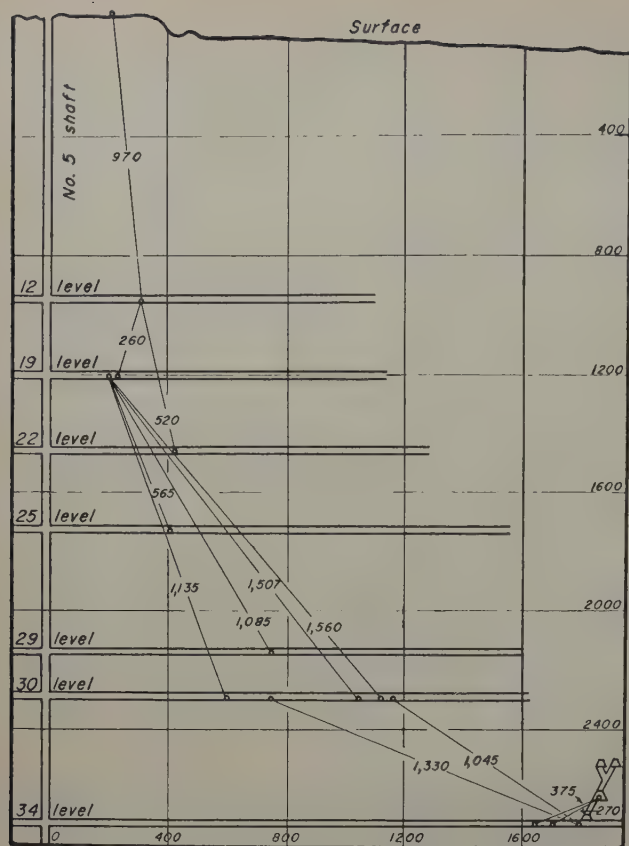


Fig. 2—Diagrammatic section showing test sites and distances, Montreal mine.

trical equipment presents a difficult problem in the Montreal mine, and it appeared that the excessive noise interference in the radio receivers was related to the grounding problems. The distance through which communication was possible between underground points appeared to be limited by noise interference rather than by lack of signal strength.

Satisfactory communication was achieved through 970 ft of soil and strata between a point on the surface and a point underground at the Montreal mine. A greater distance was possible but was not attempted because interference of the local broadcast station on the surface made two-way communication extremely difficult. Reception underground in that test was loud and clear.

Table II is a summary of results obtained in tests between underground points and between underground and surface points at the Pioneer mine, Oliver Iron Mining Division, U. S. Steel Corp., on the Vermillion iron range in Minnesota. Communication was achieved satisfactorily through 775 ft of strata between underground points. Communication was achieved through a distance of 1285 ft between underground points in one test, but some aid may

have been obtained from wire circuits in that test. Satisfactory communication was obtained through 1496 ft of soil and strata between a point on the surface and a point underground, and the low signal strengths received indicated that the distance probably could not be increased. Communication was not achieved between a point in the Pioneer mine and a point about 1000 ft away in the adjoining Zenith mine. Noise interference was excessive in that test, and faults were known to exist in the pertinent area. The reason for failure to communicate could not be determined definitely, and it was not possible to conduct tests through shorter distances between test sites in the two mines.

Communication could not be achieved satisfactorily through the strata alone at the James mine, Pickands Mather & Co., on the Menominee iron range in Michigan. Occasionally, in tests through distances not exceeding 400 ft the carrier signal was identified but no voice was heard. On fewer occasions voice was distinguishable and some words were heard on one or the other of the receivers, but two-way communication never was possible. Failure to achieve radio communication through the strata at the James mine can be attributed to peculiar strata characteristics that also interfere with geophysical investigations on the Menominee range.

Table III is a summary of test results at the Cliffs Shaft mine, the Cleveland-Cliffs Iron Co., on the Marquette iron range in Michigan. Satisfactory two-way communication was achieved through 475 ft of strata between underground points. Received signals were loud and clear, and communication could have been achieved through much greater distances, but suitable test sites devoid of wire installations were not available. Maximum distance through which communication was possible at the Cliffs Shaft mine was computed to be 1200 to 1500 ft, based on rates of attenuation determined in tests at that mine.

Table IV is a summary of test results at the Ahmeek No. 3 copper mine, Calumet and Hecla Inc., near Calumet, Michigan. Satisfactory communication was achieved through 490 ft of strata between underground points at that mine, and the maximum distance through which communication was possible was computed to be from 1250 to 1750 ft.

Frequency-response curves, made after laboratory calibration of results obtained in tests conducted in an attempt to predict rock bursts in the Ahmeek No. 3 mine, indicated some difference in results obtained in rockburst and in non-rockburst areas. Those differences, however, were not immediately discernible during the progress of the tests in the mine, even though one radio had been set up nearly 7 hr at the exact site where a rock burst occurred within 30 min after the tests were concluded.

Conclusions

1—Without dependence on aid furnished by wire circuits in the mines, voice communication by radio

Table III. Summary of Test Results at Cliffs Shaft Mine

Distance in Feet	Type of Strata	Wire Aid	Signal Strength	Noise	Remarks
240	Footwall iron formation	No	High	Low	
350	Ore, iron formation	No	High	Low	
215	Ore	No	High	Low	
475	Ore	No	High	Low	
317	Iron formation	No	High	Low	Computed maximum distance 1200 to 1500 ft Computed maximum distance 1500 feet

Table IV. Summary of Test Results at Ahmeek No. 3 Copper Mine

Distance in Feet	Type of Strata	Wire Aid	Signal Strength	Noise	Remarks
200	Amygdaloidal basalt	No	High	Low	Computed maximum distance from results of this series of tests 1250 to 1750 ft
490	Amygdaloidal basalt	No	Medium	Low	
418	Amygdaloidal basalt	No	Medium	Low	Intervening mohawkite vein caused differences in signal strengths in this series of tests
422	Amygdaloidal basalt and mohawkite vein	No	Low	Low	
254	Amygdaloidal basalt	No	High	Low	Non-rockburst area Rockburst area; frequency response differs from above after calibration
246	Amygdaloidal basalt	No	High	Low	

between all points underground or between the surface and all points underground is not feasible in metal mines in the Lake Superior district by means of currently known equipment and techniques.

2—Radio communication is possible over distances of several miles between points in proximity to wire circuits underground. Pipelines and rails near transmitting and receiving points, in the absence of wire circuits, generally do not greatly increase the range of communication.

3—Best results in communication through the strata were obtained at the lowest frequencies tested. It appears that future research in the development of a system of radio communication universally applicable in underground mines should be concentrated on the use of very low radio frequencies, ranging from 20 to 50 kc.

4—The type of rock through which signals were transmitted generally did not appear to be a controlling factor in determining the success or failure of communication, except on the Menominee iron range. No appreciable difference was observed between results obtained by transmitting through greenstone or other rock and results obtained by transmitting through ore.

5—Fault zones between transmitting and receiving points appeared to interfere with communication, and in some tests, signal strengths appeared to be diminished by passage through interfaces of contiguous dissimilar strata.

6—Noise interference and insufficient power output were the two principal factors limiting the distance through which communication was possible.

7—The communication equipment was not adaptable to on-the-spot prediction of rock bursts in a deep copper mine.

8—Additional improvements in equipment and techniques should justify future research in underground radio communication; a system of radio communication universally applicable in underground mines then may be proved feasible.

Acknowledgments

Sincere appreciation is expressed to numerous officials and other personnel of Calumet and Hecla Inc., Cleveland-Cliffs Iron Co., Montreal Mining Co., the Oliver Iron Mining Division of U. S. Steel Corp., and Pickands Mather & Co. for their wholehearted courtesy and cooperation in making possible the experiments at mines of their companies.

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Quantitative Petrographic Composition Of Three Alabama Coals

by Reynold Q. Shotts

Nitric acid oxidation rate analyses of three coals, previously studied microscopically by the Bureau of Mines, revealed three components. Relative quantities agree with those found for the four components given by the Bureau and results are consistent with current ideas of coal constitution. Possible multi-component composition for *bright* coal and a reactivity-rank relation are suggested.

THE physically dissimilar components of bituminous coals often are easily recognized megascopically. Under the microscope, reflected light or light transmitted through thin sections reveals the presence of the different components, even when these are intimately mixed. Optical methods for the quantitative estimation of the relative abundance of the various components, both by means of thin sections and by particle count, have been fully described.^{1, 2}

It has long been recognized that there are chemical and physical differences between the various petrographic components of bituminous coals, although analytical differences usually are small.^{3, 4} Only in the case of fusain have chemical differences been used for quantitative determination of a component. C. C. Hsiao and associates, at the Mineral Industries Experiment Station of the Pennsylvania State College, have described a method of analysis which is based upon the differences in the rate of nitric acid (8N) oxidation, fusain, and the other components of coal.^{5, 6} The reproducibility of their method and its applicability in checking microscopic determinations of fusain content have been supported by several independent investigations.^{7, 8}

The writer has proposed the use of differences in oxidizability for the estimation of other components.^{9, 10} The results of the oxidation of whole coals and of float-and-sink fractions of coals were reported. In most cases the plots of the logarithms of the percent dry, non-fusain, organic residue from oxidation, against time, revealed the presence of at least two distinct components. Both components appeared to oxidize according to a first order law, but the reaction constants for the components were distinctly different. One or more of the *dull* density fractions were found to contain but one component, and some of the lower rank coals oxidized in such a way as to suggest the presence of three components.

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Discussion on this paper, TP 3526F, may be sent (2 copies) to AIME before July 31, 1953. Manuscript, Oct. 25, 1951. Revised Feb. 23, 1953.

A suitable way to check the identity and significance of the components delineated by oxidation would be to analyze a sample of coal both by the nitric acid oxidation procedure and by a microscopic method. The writer was wholly unfamiliar with either of the microscopic techniques commonly used, and to make such a comparison it was necessary to rely upon microscopic analyses made by someone else. It is hoped that some laboratory which is equipped to make both types of analyses will some day make them upon identical samples.

During the past 20 years, four Alabama coals have been analyzed petrographically and the results published by the United States Bureau of Mines. They are: 1—Flat Top mine, Mary Lee bed; 2—Empire mine, Black Creek bed; 3—Wylam No. 8 mine, Pratt bed, all in the Warrior field; and 4—Soot Creek mine, Fairview bed, in the Coosa field.¹¹⁻¹⁴ Of these, only the Flat Top mine is still operating. Because of the closing of these mines, it first appeared necessary to rely upon the indirect and unsatisfactory procedure of sampling the beds in other mines located as near to the closed mines as possible. Upon investigation, however, it was found that the Bureau of Mines still had, in storage, the very same samples which had been used in the published petrographic studies. The Bureau very generously furnished about 2000 g each of the Pratt, Mary Lee, and Fairview bed coals, largely lumps but with some fines. The blocks of coal, when received, still were covered by the paraffin coating which had been placed on the polished surface, in the case of the Mary Lee coal almost twenty years ago.

Procedure

The procedure for oxidizing the coal sample and removing the alkali-soluble humic acid has been described.^{5, 9} In the present study, oxidation periods of 1/6, 1/3, 1/2, 3/4, 1, 2, 3, and 4 hr were used. All oxidations were made in triplicate. After the paraffin had been removed in boiling water and the coal washed carefully with cold benzene, the entire sample of approximately 2000 g, obtained from the Bureau of Mines, was crushed to pass a No. 4 sieve. About 200 g of this material was pulverized to pass

Table I. Proximate Analyses and Free Swelling Indexes of Coals Studied

Mine and Bed	Ash(1), Pct	Volatile(1) Matter, Pct	Fixed(1) Carbon, Pct	Sulphur(1), Pct	Heating Value Btu/Lb	Unit Fixed Carbon, Pct	Unit Heat- ing Value Btu/Lb	Free Swelling Index
Wylam No. 8, Pratt Bed	3.0	30.5	66.5	0.6	14,700	68.9	15,220	7
Flat Top, Mary Lee Bed	9.9	27.6	62.5	0.6	13,660	70.1	15,310	8
Soot Creek, Fairview Bed	7.1	27.6	65.3	2.2	13,850	71.3	15,510	7½

(1) Dry basis.

a No. 200 sieve and stored in bottles. As oxidations were made, a few grams were removed from the storage bottle, placed in a weighing bottle, and dried. All analyses were run using dry coal. Since the petrographic sections given in the reports were uniform, it is likely that the blocks of coal furnished by the Bureau were genuinely representative of the entire bed section.

Chemical and Physical Properties of the Coals

Analyses and free swelling indexes of the three coals are shown in Table I. Heating values and free swelling indexes undoubtedly have been reduced by the long storage period, although the present values indicate that only small changes have occurred.

The three coals are almost identical in rank as shown by the values for dry fixed carbon free of mineral matter. Pratt coal falls very near the border line between medium-volatile bituminous and high-volatile A bituminous coals. The other two coals are just inside the medium-volatile bituminous group. The non-fusain portion of Fairview coal may not be of quite the rank indicated because of the relatively high fusain content. The presence of considerable fusain in this coal probably reduces its free swelling index slightly.

Results of Oxidations

Fig. 1 shows the logarithm of the percent moisture and ash-free residue, minus the fusain residue, plotted against the time. Fusain residues were read from Fig. 2. It is evident that for all three coals oxidation was rapid during the first 10 min. A straight line has been drawn connecting the non-fusain organic matter at zero time with that remaining at 1/6 hr, indicating that the full 10-min period was required for the oxidation of the more reactive portion of the coal and that the rate was uniform for the period. This may not have been true, but because of the absence of shorter oxidation periods it was necessary to make this assumption. A reaction rate constant based upon this assumed straight line is not

an accurate value but should represent an approximate average for the first 10 min.

It will be observed that only in the case of the Mary Lee coal did the percent of residue, after 2-hr oxidation, fall near the line drawn through the other points. This is not surprising, for the non-fusain residue, after such prolonged oxidation, was very small for all the coals. Slight analytical errors could result in sizable displacement from the true value.

Fig. 2 shows the longer oxidation periods plotted against time, on rectangular coordinates. Extrapolation to zero time of the line joining percents residue after 3 and 4-hr oxidations gives the percent of fusain in the sample.⁵

Comparison with Microscopic Analyses

Table II shows the composition of the three coals as published by the United States Bureau of Mines and as determined by oxidation with nitric acid. The table also shows the specific reaction constants for the *bright*, *dull*, and fusain components.

The symbols used in Tables II to IV are the same ones used previously by the writer and others.^{5, 9} The symbols C_{e-1} , C_{e-2} , C_{e-3} , C_f refer to the percent of fusain and of other coal components, in descending order of their brightness, in the original dry, ash-free coal; K_{e-1} , K_{e-2} , K_{e-3} , K_f are the corresponding specific reaction constants for the same components, in units of percent of material oxidized per hr.

In the case of Pratt and Mary Lee coals, the percent of dull coal, fusain-free basis, was larger than that of the Bureau of Mines' opaque attritus and less than the sum of the opaque and the translucent attritus. For Fairview coal, the dull component was equal to the sum of the translucent and the opaque attritus. The percentages of fusain, as determined by the two methods, did not check exactly, although it is probable that the values shown differ little, if any, from those to be expected for duplicate samples of the same coal as determined by either method. The samples sent by the Bureau of Mines were largely of lump coal, so that the fusain content ob-

Table II. Comparison of the Results of United States Bureau of Mines Microscopic Analyses with Oxidation Analyses Using 8N Nitric Acid, Dry Basis and Dry, Fusain-Free Basis

Bed	United States Bureau of Mines				Nitric Acid Oxidation Method					
	Anthraxylon, Pct	Translucent Attritus, Pct	Opaque Attritus, Pct	Fusain, Pct	Bright, Pct C_{e-1}	Dull, Pct, C_{e-2}	Fusain, Pct, C_f	Specific Reaction Constants, Pct Per Hr		
								K_{e-1}	K_{e-2}	K_f
Pratt	51	36	8	5	77.2	16	6.8	12.62	1.28	0.82
Mary Lee	61	27	5	7	68.0	26	6.0	9.87	2.55	0.65
Fairview	57	20	8	15	58.8	29	12.2	8.25	1.66	0.95
On Dry, Fusain-Free Basis										
Pratt	54	38	8		83	17				
Mary Lee	66	29	5		72	28				
Fairview	67	24	9		67	33				

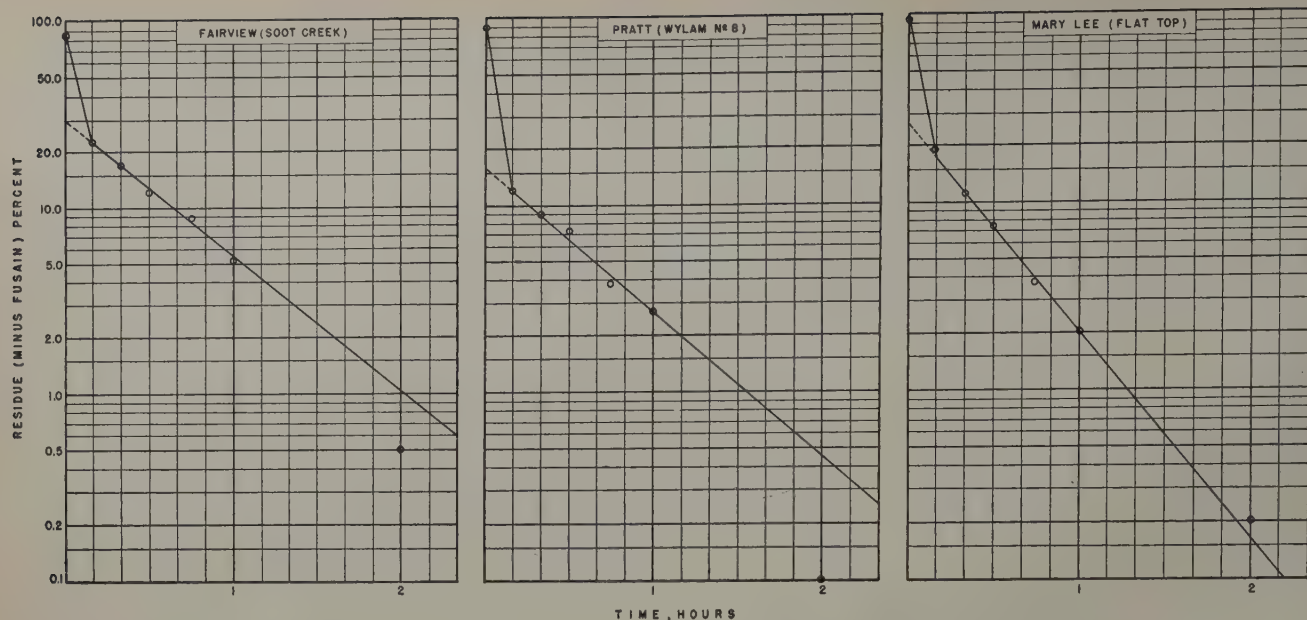


Fig. 1—Logarithm of percent total residue, less fusain residue, as a function of time of oxidation for three Alabama coals.

tained could be expected to be slightly smaller than for the entire column originally studied by the Bureau. It will be noted that the chemical method gave the expected smaller values for Mary Lee and Fairview coals but a larger one for Pratt coal.

When the experimenter attempts to determine the extent to which the components delineated by dif-

Table III. Comparison of the Results of Oxidizing Lewisburg Mine, Mary Lee Bed, Coal* with 8N and 4N Nitric Acid

Conc.	C_f	C_{c-1}	C_{c-2}	C_{c-3}	K_f	K_{c-1}	K_{c-2}	K_{c-3}	Unit F C
8N	4.8	60.2	35.0		0.17	9.09	1.75		69.4
4N	4.1	47.9	32.0	16.0	0.07	5.58	0.65	0.21	69.4

* Float 1.80 — sink 1.34 fraction of washed coal.

ferences in oxidation rates correspond to those identified and estimated by microscopic study, he is handicapped by a lack of knowledge of the chemical differences, if they exist, between the microscopic components. From the general descriptions given of petrographic components and their properties,^{3, 4} it appears certain that when specific gravity separa-

tions of coal are made, the fractions of lower density will contain some concentration of anthraxylon and of bright attrital coal, while the heavier fractions will contain most of the dull coal, or opaque material, and the fusain. Previous work^{9, 10} on Alabama coals has shown that the lighter density, or bright, fractions are much more readily oxidized than the heavier, or dull, ones. For this reason the more reactive component, C_{c-1} , revealed by the oxidation analyses must include the greater part of the bright components, and the less reactive fraction, C_{c-1} , the dull or opaque components. As shown above, the fusain components determined by the two techniques appear to correspond fairly well.

If, as the nitric acid oxidations indicate, one component oxidizes with relative ease and another with relative difficulty, the first component must include at least the anthraxylon of the optical components and the second one at least the opaque matter of the attrital optical components. If this much is admitted, the translucent attritus, or translucent humic matter, of the optical analyses must contain some material relatively easy to oxidize and some relatively difficult to oxidize. It is not impossible, of course, that some of this material actually is of intermediate oxidizability. If such material exists, it must have been included, for the three coals of this study, in the portion which oxidized during the first 10 min of boiling C_{c-1} . Some evidence has been published of the existence of a third component in some low rank coals and even of a component intermediate between fusain and dull coal in the densest fraction of a medium volatile coal.⁹ The results shown in Table III and Fig. 3 indicate three components when weaker 4N nitric acid was used for oxidations. The log percent non-fusain coal vs time plots of Lewisburg mine coal, Mary Lee bed, as oxidized with 8N nitric acid, shows no evidence of a third component. Whether or not the intermediate component was equivalent to translucent humic matter could not, of course, be determined. The great rapidity of oxidation with 8N nitric acid may mask the presence of matter of intermediate reactivity.

If as seems likely, however, there are normally only two principal components present in addition to fusain, and these components correspond roughly

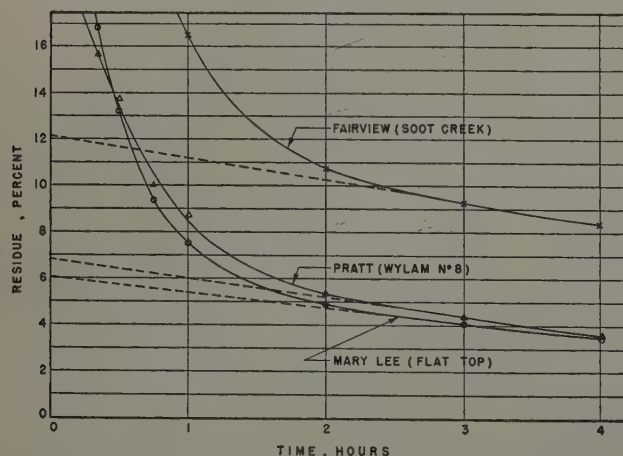


Fig. 2—Plot showing the percent total residue, after prolonged oxidation, as a function of time.

to bright coal and dull coal, it would be expected that the sum of the anthraxylon and the translucent attritus of the Bureau of Mines analyses would be larger than the bright or easily oxidized chemical component and the sum of the translucent attritus, and the opaque attritus of the Bureau of Mines would always exceed that of the more difficultly oxidized chemical component. As noted above, this was the result found in the case of all three of the coals studied. The existence of three such cases is necessary, if not sufficient, evidence for establishing a relationship between the chemical components and those differentiated and estimated optically by the Bureau of Mines.

Specific Reaction Constants

The specific reaction constants for the corresponding bright K_{c-1} and dull K_{c-2} components were very nearly the same for the three coals, as expected from the similarity of rank. The contrast between reactivity of bright and dull components for a given coal was greatest for Pratt coal and least for Mary Lee coal. For any use as a chemical raw material, such as for hydrogenation, removal of the dull component would improve Pratt coal more than it would the other two. The dull component of Fairview coal is only slightly more reactive than that of Pratt coal but it makes up twice as much of the sample.

It might prove desirable to rate coals according to their overall chemical reactivity. The case just cited, of Fairview and Pratt coal, illustrates how difficult it may be to decide which coal really has the greater overall chemical reactivity. One way to do this would be to multiply the percentage of the bright component by its specific reaction constant, the dull component by its specific reaction constant, and find the sum. Fusain could be included, but its influence on the magnitude of the sum would be insignificant. Such a weighted, overall reactivity factor or index would have units with no physical meaning. Absolute values would be meaningless also, but an index of this kind should be of value for comparing the relative overall reactivities of similar coals, oxidized under the same conditions.

Table IV shows such indexes calculated for the three coals. The dull component of each coal contributes a relatively small fraction to an index calculated in this way, in the case of Pratt coal less than three percent. Microscopic analyses yield no comparable quantitative measure of reactivity or of a correlate property.

Table IV. Overall Reactivity Indexes of the Three Coals Studied

Bed	$C_{c-1} \times K_{c-1}$	$C_{c-2} \times K_{c-2}$	Overall Reactivity Index
Pratt	1047	31	1078
Mary Lee	711	71	783
Fairview	553	55	608

It will be noticed from Tables I and IV that the overall reactivity indexes are in the inverse order of the ranks of the coals. The differences in rank are so small, however, that little significance can be attached to this result. Fusain content may have influenced the rank of the coals but not the overall reactivity indexes. Comparison analyses of coals differing decidedly in rank are needed before any final

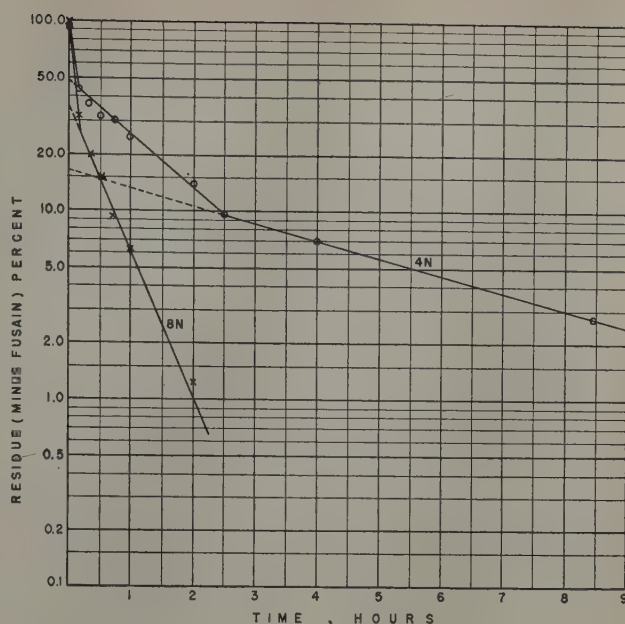


Fig. 3—Logarithm of percent total residue less fusain residue, as a function of time of oxidation by both 4N and 8N nitric acid for Mary Lee Bed, Lewisburg mine.

conclusions regarding rank-reactivity relations can be drawn.

The specific reaction constants for fusain differ slightly and increase a little with increasing quantities of fusain material present. This result suggests that fusains differ somewhat in reactivity and that apparently the coals higher in fusain contain the larger quantities of the more reactive type. At least two physically different types of fusain have, in fact, been identified in microscopic studies.⁵

Bearing on Problems of Origin and Constitution

There is little doubt that the differences in oxidation rate constants encountered with the various components of the same coals, and in coals of different rank,⁹ throw some light upon the geological problems of coal origin and the nature of the factors determining rank, as well as the chemical problem of coal constitution.

Regarding the latter problem, two distinct views seem to prevail. The older one, apparently held by most workers in Great Britain as well as by many in this country, has been expressed by Lowry: "The coal molecule has resulted from condensation and polymerization of polynuclear six-membered carbon ring compounds containing hydrogen, oxygen, nitrogen, sulphur, and other elements found in coal. It is suggested, on the basis of the nature of the products obtained from the mild oxidation of coal, that an important unit of structure has essentially the same nuclear structure as humic acids."¹²⁵ Another view of the molecular structure of humic acid and, it may be inferred, the humic portions of coal, has been stated by Kinney, Polansky, and Guager: "The behavior of humic acids . . . has given credence to the fact that humic acids derived from nitric acid-treated bituminous coal are not totally characterized by a polycondensed ring structure but rather one in which the nucleus is more or less in a carbonized state approaching the character of 'amorphous' or graphitic carbon."¹²⁶

Difference in rate of reaction with nitric acid may be explained by either picture of the dominant mole-

cule. If many of the molecules in coal material are condensed polynuclear substances, rate of nitric acid oxidation should be determined by the amount of condensation. Presumably higher rank coals or petrographic components are more highly condensed than those of lower rank and should therefore exhibit smaller reaction rate constants. Reactivity with nitric acid follows this order. On the other hand, if many molecules in coal are partly graphitized, the oxidation rate should decrease with degree of increased graphitization. This too seems to correspond with the facts.

It is difficult to account for differences in *dullness* and *brightness*, or for markedly different rates of reactivity with nitric acid and other chemical reagents shown by intimately mixed petrographic components which have been subjected, in any particular locality, to the very same dynamical agencies. The explanation may be 1—that the components came from unlike plant material or 2—that they were subjected to different chemical, biochemical, or physical forces previous to burial. One or the other of these factors has usually been resorted to for explanation of differences in chemical composition, physical appearance, or physical properties. Undoubtedly differences in reactivity toward nitric acid shown by physical components can also be traced to one or the other of these factors.

Summary

The results of this study may be summarized as follows. 1—The non-fusain portion of the coals studied consists of two components of differing reactivity toward nitric acid. 2—The *dull* component of each of the coals includes the greater part of the material in the component called *opaque attritus* and part of the material known as *translucent attritus*, to use terms employed by the Bureau of Mines; the *bright* fraction contains the anthraxylon and the remainder of the translucent attritus. 3—In one case oxidation with nitric acid of lower concentration revealed the presence of an intermediate component. This component may have represented material truly of intermediate oxidizability. It is possible that a concentrated oxidizing acid acts upon the condensed or graphitized coal structure so strongly that molecules of intermediate order of condensation or graphitization do not show rates significantly different from those of lower order. 4—The percentages of fusain, as determined by the nitric acid oxidation method and microscopic studies made by the Bureau of Mines, were in fair agreement. 5—Specific reaction constants for corresponding coal components were similar, a result expected because of the similarity in rank of the coals. 6—An arbitrary, overall reactivity index calculated from the values determined from the oxidation analyses apparently correlated with rank, but rank differences were too small for this result to be of significance. 7—The two-component nature of the non-fusain portion of the three coals, as revealed by oxidation procedures, is not inconsistent with widely accepted ideas of coal constitution and origin and rank changes in coal.

Acknowledgments

The major portion of this work was made possible by a grant from the University of Alabama Research Committee.

Many of the proximate analyses and heating values and a few oxidations were made by Lamar

Campbell and J. D. Cowen, student analysts. H. G. Smith, graduate student at the University of Alabama, made the 4N nitric acid oxidations of Lewisburg coal. The author wishes to acknowledge the able and timely assistance of these men.

Grateful acknowledgment is made also of the splendid cooperation of Dr. A. C. Fieldner, Dr. Bryan Parks, Dr. Ralph L. Brown, and other United States Bureau of Mines officials who furnished the samples of coal.

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The Role of Geologists in the Development Of the Labrador-Quebec Iron Ore Districts

by J. K. Gustafson
and A. E. Moss

MEASURED in terms of ore tonnage, the New Quebec and Labrador iron ore fields promise to rank with the greatest iron ore districts of the world. Over 400 million tons of high-grade ore for direct shipping have been discovered to date. Much greater amounts of lower-grade beneficiating ores are indicated by outcrops. Many hundreds of square miles of iron formation await testing.

By 1954 iron ore will begin to move towards American blast furnaces, and shortly thereafter the annual rate of production is expected to reach 10 million tons. To bring this vast project into being over \$200,000,000 will have been spent for exploration and development, mining equipment, hydro plants, townsites, the building and the equipping of a 360-mile railroad, and the construction of docks and ore-handling facilities.

The most important iron ore deposits are located over 300 miles from tide water, see frontispiece and also Fig. 1. The frontispiece shows principal sources of iron ores and manganese iron ores for the U. S. and Canadian steel industry, and the major smelting districts are shaded on the map. No navigable rivers lead into the interior. When work was commenced on this project there were no roads, no railroads, no communication system, no inhabitants, and no adequate maps either topographical or geological. The geological field season was limited to approximately 100 days per season.

The obstacles facing this enterprise were many and varied. Its success will be due to the integrated team work of people of diverse skills. Explorers, Indian guides, prospectors, geologists, surveyors, mechanical, civil, electrical, and mining engineers,

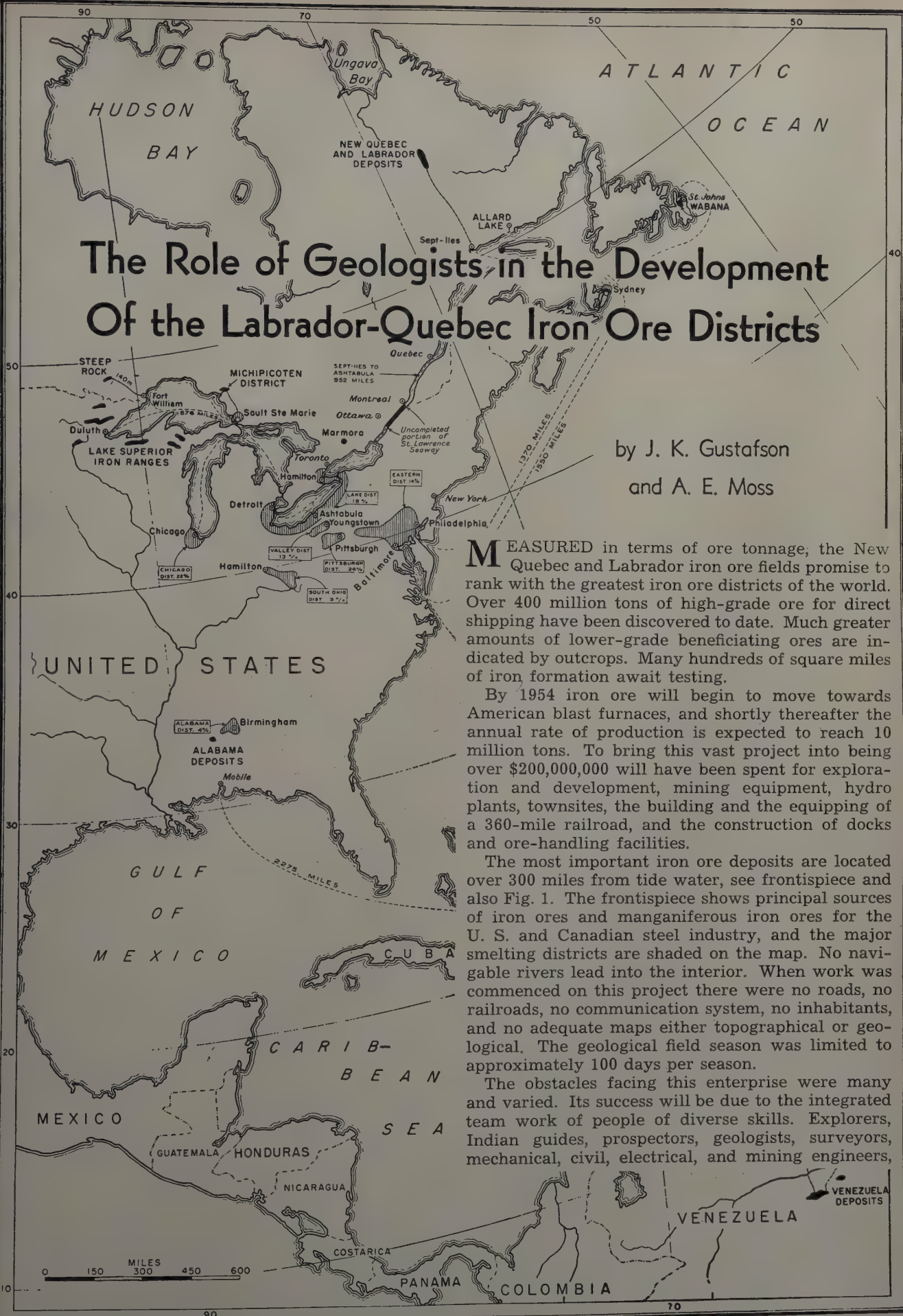




Fig. 1—Principal sources of iron ores and manganiferous iron ores for the United States and Canadian Steel industry.

railway experts, construction experts, drillers, mechanics, aeroplane pilots, cooks, executives, and financiers have all participated.

Hollinger Consolidated Gold Mines Ltd. is a major shareholder and Hanna Coal and Ore Corp. a minority shareholder in two companies, Labrador Mining and Exploration Co. Ltd. and Hollinger North Shore Exploration Co. Ltd., which have exclusive prospecting rights in two large concessions totaling nearly 25,000 square miles in Labrador and Quebec. Ultimately, these concession companies will lease much smaller areas from the Provincial Governments and will each sub-lease still smaller areas to Iron Ore Co. of Canada, a company formed by the same interests in association with five American steel companies for the purpose of bringing this project into production. The American companies are National Steel Corp., Armco Steel Corp., Republic Steel Corp., Wheeling Steel Corp., and Youngstown Sheet and Tube Co.

Within the next few years nearly 95 pct of the concession areas will be returned to the Provincial Governments. Geological work has had two main purposes: 1—to find and develop enough ore to justify bringing this project into production, and 2—to learn enough about the concessions to permit selection in the time available of the best 500 square miles from an iron ore standpoint for sub-leasing. The total areas being investigated are approximately

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Discussion on this paper, TP 35531, may be sent (2 copies) to AIME before Aug. 31, 1953. Manuscript, Feb. 3, 1953. Los Angeles Meeting, February 1953.

the size of a strip of land 10 miles wide joining New York and Los Angeles.

The object of this paper is to describe this geological effort. That much correlated geological knowledge exists will have to be taken on faith until it is published elsewhere. The staff of the Iron Ore Co. of Canada are now engaged in compiling a detailed report summarizing the geological data gathered by the vast number of geologists who have been associated with this project since 1936. Probably this report will be published in a few months. Sufficient description of geology and topography will be given here to make the paper understandable.

The area under discussion is a glaciated plateau of moderate relief, see Fig. 2. Most of the area is between 1500 and 2500 ft above sea level. Lakes and rivers are numerous, and water covers over 50 pct of the surface throughout large areas. Much of the south-central section is covered by marshes and swamps. The distribution of the vegetation is erratic. In valleys and sheltered areas there are fair stands of black and white spruce with some birch, balsam, and tamarack. Higher areas are above the timber line and are carpeted by white caribou moss. Patches of shrubs grow along slopes and in ravines.

Rock outcrops throughout the northern half of the area are numerous and widespread, although vast continuous rock exposures such as are found in the Northwest Territories are absent. Outcrops throughout the southern portion are chiefly confined to the crests or slopes of the larger ridges. Here, geological features could not be determined by mapping alone. Test pitting, drilling, and/or geophysical methods were also employed.

Although the center of the Labrador ice cap was near this area, evidence of severe glacial erosion is

lacking. The area is now covered with an irregular, generally shallow mantle of glacial till. In the southern portion thick deposits of gravel and sand are present. Eskers are numerous.

The consolidated rocks are pre-Cambrian. They have been intensely folded and faulted. Many folds are overturned, and abrupt changes in direction and degree of plunge are common. The regional strike is generally northwest-southeast or north-south, but east-west folds and even northeast-southwest structures occur in the southern portion of the area. Strike faults, mostly thrusts, are numerous. Most faults are steep. Small cross-faults occur.

The main ore zone, in which most of the known ore exists, trends northwest-southeast. It is approximately 80 miles long and averages about 6 miles wide. Table I lists the formations of rocks of the main ore zone and vicinity. Fig. 3 shows the surface geology and two vertical cross-sections of a typical area within the main ore zone.

The host rocks of the iron ores are the Sokoman iron formation and the Ruth slate, although in several places the iron ores extend a short distance into the underlying Wishart quartzite. All of the iron formation probably belongs to one stratigraphic horizon, although it is repeated numerous times by folding and faulting. Locally it is intercalated with lava flows and fragmentals of intermediate or basic composition. Unenriched Sokoman iron formation

averages about 30 pct iron. It contains iron oxides, iron silicates, iron carbonates, and cherts (locally recrystallized to quartz). Eight to ten rock types within the Sokoman are recognized for mapping purposes. Most rock and ore types can be duplicated in iron ranges of the Lake Superior region.

The ore varies greatly from one orebody to another and even from place to place within individual deposits. Most of the ore consists of mixed hematite and geothite. Deposits of almost pure hematite are known, and either geothite or limonite may occur alone. Martite is usually abundant. Occasionally magnetite is present. To date, siderite has not been observed within an orebody although it is prevalent in lean iron formation in some places. The ore ranges from hard and dense to soft and porous. The moisture content varies widely. Approximately 10 pct of the ore is manganiferous and occurs as lenses within the iron ore.

The iron ore of the main ore zone is essentially shallow and is believed to have resulted from the weathering of low-grade iron formation. The structural control of ore deposition is not everywhere clear, although certain structures including synclines were favored during ore formation.

In the southern portion of the Labrador concession a higher grade of metamorphism prevails. Large masses of iron formation averaging over 40 pct iron have been recrystallized to a granular magnetite-

Table I. General Table of Formations for Iron Ore Areas

CENOZOIC		{	Glacial, lake, and stream deposits		
Unconformity					
PROTEROZOIC	MONTAGNAIS (Keewanawan?)	{	* Diabase, diorite, gabbro, dikes, and sills Sims Group: Pink and white quartzite		
	Unconformity				
	KANIAPISKAU (Huronian?)	Point Series	{	Menihok Formation Black and grey shales; dolomites	
		Unconformity			
		Ferriman** Series	{	Sokoman Formation Iron formations; ferruginous cherts and slaty members	
				Ruth Formation Rusty, black ferruginous slates	
				Wishart Formation Quartzites	
		Unconformity?			
		Hamilton** Series	{	Fleming Formation (restricted occurrence) Chert breccia	
				Denault Formation Dolomites	
				Attikamagen Formation Grey and green shales	
				Seward Formation Quartzites and grits	
Unconformity					
ARCHEAN		{	Ashuanipi Complex: Orthogneisses-paragneisses Acidic and basic intrusives		

* In the southwestern corner of the Labrador concession the Kaniapiskan rocks are cut by granites and other acidic intrusives.
** In the eastern portion of the area, these formations are intercalated with lavas, sills, related pyroclastics, and greywackes.



Fig. 2—A photograph showing the terrain of the Labrador-Quebec iron ore district. An airstrip lies near the railroad right of way, which can be seen following the river valley.

specularite-quartz rock which readily makes a high-grade concentrate by gravity separation. Whether this iron formation is richer because of initial differences in sedimentation or because of enrichment prior to recrystallization is not yet known.

History of Geological Work

Geological Investigations Prior to 1936: Prior to 1936 little geology was known. Until the 1890's the few white men penetrating the interior were missionaries and fur traders employed by Hudson's Bay Co. The first geological investigations were conducted by A. P. Low of the Geological Survey of Canada¹ in 1892 to 1895. Low did a monumental job of reconnaissance mapping along several traverses through the interior and along the coast of the Ungava Peninsula. He traced for several hundred miles the great belt of Huronian-type sediments since termed the Labrador Trough. He discovered the iron formation and suggested that the area was favorable prospecting ground for iron ore because its geology was similar to that of the Lake Superior region. During 1892 to 1895 Low travelled approximately 4500 miles by canoe, 1000 miles by boat, 500 miles by dog sled, and 1000 miles by foot, a total of 7100 miles.

In 1929 W. F. James and J. E. Gill² conducted an expedition for the New Quebec Co. which discovered the first iron ore in the region. The five geologic teams of this expedition mapped ore showings and made the first rough topographic maps compiled from sketches made during aeroplane flights. In 1933 J. E. Gill led an expedition into the Wabush Lake area which is now part of the Labrador conces-

sion.³ This party, including two other geologists, was organized to examine auriferous quartz veins which had been reported.

The only geological information available in 1936 came from these two private expeditions and one reconnaissance traverse made by Low's party. With the exception of a few sketches, there were no topographic maps. Although the Quebec government had made a stadia traverse along the Ashuanipi River, this information had not been released. No aerial photographs existed.

Geological Investigations from 1936 to 1944: In 1936 Labrador Mining and Exploration Co. was formed, and a concession of over 20,000 square miles was obtained from the Newfoundland government. J. A. Retty was appointed chief geologist. Field operations were based at Sandgirt Lake during the following four seasons. A major problem was the lack of base maps. Sketch maps made from aeroplanes were used to plan the work and to distribute and relocate field parties. Rapid stadia traverses along the main water courses were made during the next three years supplemented with range-finder and compass traverses. From 1936 to 1939, 12 two-man geological parties mapped a considerable area using reconnaissance methods. The stratigraphy and petrology of the region were beginning to unravel by the end of 1939. Iron ore was found in Quebec along the strike of the 1929 discoveries. In 1937 the Sawyer Lake orebody was shown to Retty by an Indian. This deposit was trenched and mapped. By the end of 1939 six of the currently recognized orebodies were found. Work was suspended during 1940 and 1941 because of a shortage of money.

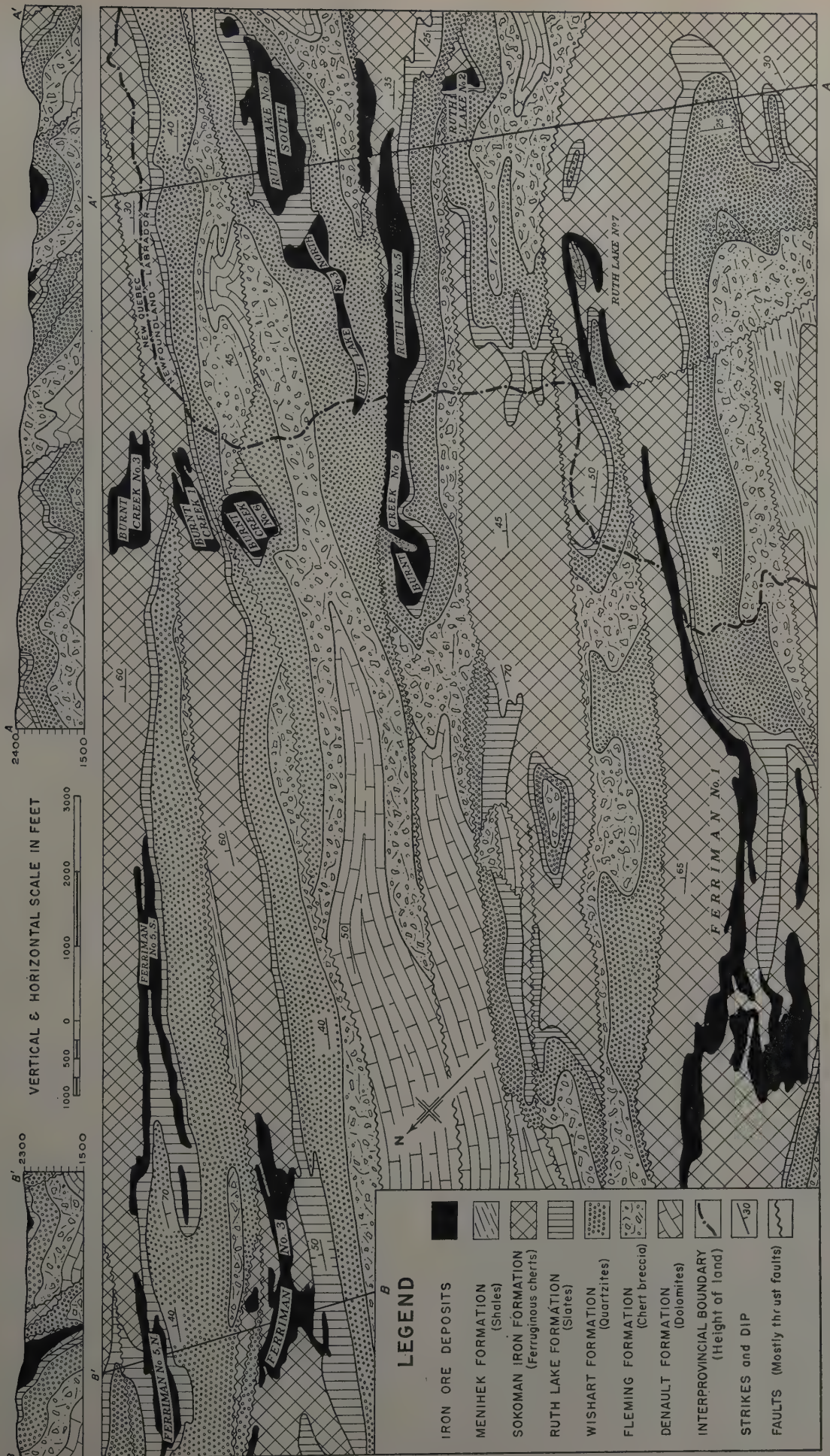


Fig. 3—Geology and iron ore deposits of a portion of the main ore zone.



Fig. 4—A churn drill operating on one of the Fleming ore deposits. Note barren nature of territory.

In 1941 Hollinger Consolidated Gold Mines Ltd., following the recommendation of J. K. Gustafson, obtained an option to purchase control of the Labrador Mining and Exploration Co. Ltd. Hollinger also acquired exclusive prospecting rights on an adjacent area of 3900 square miles in New Quebec on the recommendation of Retty. A new company, Hollinger North Shore Exploration Co. Ltd., was formed to hold this concession. The first Hollinger-controlled geological and prospecting expedition in 1942 was headed by Gustafson with Retty second in command. A. E. Moss was also a member of this party. Aerial photographs being unobtainable during the war, maps sketched from aeroplanes and range-finder and compass surveys were again used, one for planning work and the other for preparing base maps. The principle now in use of mapping observed outcrops along plotted traverse lines was established. Professional prospectors were put into the area. A regional geological map showing a structural and stratigraphic interpretation based on data available at that time was prepared. Iron ore showings were re-examined, but the work during 1942 was concentrated on the search for metals other than iron. More than a hundred large gossans resulting from oxidation of sulphides were superficially examined, but no commercial orebodies discovered.

In 1942 to 1943, the M. A. Hanna Co. purchased an interest in both exploration companies. Thereafter the Hanna technical staff, including A. E. Walker and Mack C. Lake to 1949, acted in a consulting capacity. During 1943 and 1944 prospecting, detailed and reconnaissance geological mapping, and some drilling were continued under the field direction of J. A. Retty. Several diamond drill holes were drilled into some of the sulphide deposits, and a few holes were drilled into the Sawyer iron ore deposit. Twelve deposits containing 160,000,000 tons of iron ore were discovered by prospectors during this period. The first government aerial maps of the area became available. In 1943 the base of field opera-

tions was moved from Sandgrit to Hollinger Lake.

Geological Investigations, 1945 to 1950: Prior to 1945 the exploration consisted principally of semi-reconnaissance geologic mapping and prospecting for nonferrous base metals. In 1945 exploration for iron ore began in earnest in response to the growing realization of postwar needs for new reserves. Detailed mapping and test pitting of known iron ore showings were begun. The surface area of ore was increased many times during 1945 and 1946. This increase resulted mostly from extending the surface area of previously discovered orebodies, although several new deposits were found. In 1946 iron ore deposits in the main ore zone were drilled with diamond drill rigs. Small tractors were flown into the area the same year. The first vertical aerial photographs of the area were taken in 1946 by R.C.A.F.

During the winter 1946 to 1947 an objective was set of proving 300 million tons of direct shipping, open-pit ore as a prerequisite for financing to the production stage. Intensive drilling operations to determine the depths of the large surface area of ore began. The field base was moved from Hollinger Lake to its present site near Knob Lake. Quonset buildings were flown into Knob Lake to house exploration personnel. A portable sawmill was delivered so that buildings could be erected from local timber. A semi-permanent camp was constructed including bunkhouses, dining hall and kitchen, office, warehouses, chemical laboratory, and a modern machine shop. Although most of the supplies and equipment in 1947 were landed on the winter ice of Knob Lake, a land strip was put in operation by early fall so that large aircraft could land the year round. Heavier tractors were flown in and began the first roads. Trucks, jeeps, compressors, and drills followed. Portable churn drills and equipment for converting the unsatisfactory coring drills to chopping drills were purchased, see Fig. 4. The first large caterpillar-mounted churn drill arrived in the early fall of 1947. Three more followed in 1948. Development of air strips and railroad lines proceeded simultaneously, see Figs. 5 and 6.

The 300 million ton objective was reached in October 1948, but by then the requirements had been raised to 400 million tons. By the end of 1950 ore reserves reached the present figure of 418 million tons of direct shipping ore. Over 2700 tons of ore were indicated with each foot of drilling. This high rate of yield was largely due to the concentration of effort resulting from having accurate, detailed geological maps. Thus waste of footage in barren sections was kept at a minimum.

Drill cuttings were used almost exclusively for analyses and ore calculations as it was found impractical to core the ore. Separate samples were collected for each 5-ft section. One to two pounds of each sample were stored for future reference after the sample had been logged and analysed. A binocular microscope proved to be indispensable for determining the mineral and textural composition of the ores and other formations. With it the iron content can usually be estimated within 2 pct.

Most but not all effort during the 1947 to 1950 period was devoted to drilling. In 1947 adits were driven into two of the ore deposits, and three shafts were excavated the following year. These underground workings were driven to check the ore grade as determined from drilling and to obtain information as to structure, specific gravity, and moisture content of the ore.

Much pitting and trenching through the overburden were done. Previous to 1949 this work had been done with hand tools or tractor blades. Small mechanical shovels equipped with a back-hoe attachment were obtained that year capable of digging to a depth of 15 ft. These proved to be tremendously important tools for outlining orebodies.

The specific gravity of the ore received attention during this period. Because large tonnages and great variations in the character of the ore were involved, the problem was complex. Volume factors were determined from the underground workings by weighing all the material excavated and by measuring the volume of the excavation by taking offsets from a template. Although accurate, the results were valid only where there were expensive adits or shafts. After many tests the following procedure was adopted. The overburden was removed to expose the ore over an area of several square yards. The surface of the ore was then levelled and a 2 to 3-cu ft excavation made in it. All the material removed was carefully weighed and the volume of the excavation determined by sealing the walls and bottom of the pit with a rubber-like plastic applied with a spray gun and filling the pit with a measured quantity of motor oil. This method is believed to be superior to those using shot or grain as all irregularities are filled with oil and there is no compaction. The method is inexpensive and quick.

Ground magnetometer and gravimetric surveys of selected areas were made in 1947, 1948, and 1950. The Canadian government and the company established triangulation stations throughout much of the area. Accurate planimetric base maps were prepared from aerial photographs using the triangulation stations for ground control. The company established its own photogrammetric section and has prepared base maps of all potential iron ore areas using vertical aerial photographs taken from company aircraft on a scale of 1 in. to 1000 ft.

In 1950 the amenability of marginal ores to beneficiation was investigated. It was found that large tonnages of wash ores are available. Many screen tests were made.

In 1947 and 1948, and to a lesser extent in 1949, large areas along the main ore zone were mapped on a scale of 1 in. to 200 ft using plane tables. In 1949 the program of mapping all areas underlain by iron formation on 1000 ft to the inch was commenced.

Outcrops and traverses were plotted on the vertical aerial photographs, and additional notes were recorded on transparent plastic overlays. Each area was systematically traversed, and all outcrops of iron formation were located and classified. In 1949 and 1950 mapping on a scale of 1 in. to half-a-mile and some reconnaissance mapping were resumed after having been discontinued in 1946 and 1947.

Geologists directed a large part of the ore development program of 1945 to 1950. They spotted all drill holes, adits, and shaft sites. They logged all drill holes. Geologists supervised all sampling of drill holes and underground and surface workings. Volume factor and structure tests were under their jurisdiction. They calculated ore tonnages and grades and drew sections and plans prerequisite to these calculations. This work was supervised by A. E. Moss, who was assisted by R. Geren and R. Kirkland, who have recently joined the operating department.

In 1949 five American steel companies together with the Hollinger and Hanna interests formed Iron Ore Co. of Canada, which arranged to sublease areas from the two concession companies. Financing and a full-scale construction program were assured in 1950. Thus a new phase of this project began, although the work continued under the same management. In the same year J. M. Harrison of the Geological Survey of Canada started geological investigations in the area.

Geological Investigations, 1951 to 1952: Once the decision to proceed with railroad construction and iron ore production was made the finding of ore was no longer urgent. Emphasis was now on construction and procurement of equipment and supplies.

A mining department was organized, and several of the geological personnel were transferred to it. Drilling continued primarily to determine pit outlines and to locate screening plants, dump areas, and spur lines off of ore. Additional shafts and trenches were excavated, and many more samples were collected for wash and screen tests. Five-foot aerial contour maps covering over 30,000 acres were prepared to aid in laying out spur lines and mining facilities. The first ore deposits to be mined were mapped on a scale of 1 in. to 100 ft.

The geologists' main problem became one of evaluating large areas underlain by iron formation in order to select the best 500 square miles of territory



Fig. 5—Airstrip at Mile 36 along Q.N.S. & L. railway, one of 13 built for railroad construction. A DC-3 is being unloaded. The rugged country is typical of the first 100 miles north of Seven Islands. Fig. 6—A railroad rock cut along right of way.

for subleasing by Iron Ore Co. of Canada. Triangulation surveys were continued, and thousands more aerial photographs were taken. The 1000-scale mapping program was accelerated, and by the end of 1952 all areas of maximum interest had been mapped on this scale. A few new ore deposits were found but not drilled. An aeromagnetic survey was made of areas of sparse outcrops and of a strip along the railroad route. Approximately 20,000 line miles were flown during this survey. Drilling in 1952 tested magnetic anomalies in drift-covered areas.

The Geological Survey of Canada has published preliminary maps of certain areas investigated since 1949.⁴⁻⁶ The company has worked in close conjunction with the Survey and has made available to it all company geological data. Harrison published a report on a portion of the district in 1952.⁷

In addition to the aeromagnetic surveys along the railroad route, other geological investigations were conducted outside of the concession areas. Reconnaissance and detailed mapping were done of selected areas adjacent to the railroad route. Iron-bearing beach sands were examined at certain points along the St. Lawrence River.

Future Geological Investigations: The broad geological features of a large territory have been roughed out, and detailed data have been assembled in the vicinity of known ore deposits. Much, however, remains to be done. Geological investigations will continue and iron ore discoveries will continue to be made for many decades. In addition to iron, other metals will require investigation both within and outside the concession areas. Recent detailed drilling has already yielded surprises requiring modification of geological structural interpretations.

Iron Ore Co. geologists will continue 1000-scale mapping and will map selected areas on larger scales. Trenching, pitting, and drilling will follow or accompany detailed mapping. Areas containing

beneficiating ores will be re-studied. This spring several geologists will have a conducted tour through pits, plants, and laboratories on the Mesabi and Cuyuna ranges to learn more about wash, heavy media, and other beneficiating ores.

Organization of Geological Work

The year-round staff of the Iron Ore Co.'s geological department consists of a chief geologist, A. E. Moss, who succeeded J. A. Retty in January, 1952; 2 supervising geologists, C. Dufresne and H. E. Neal; 6 other geologists; 4 surveyor-topographers; 4 draftsmen; and a clerical staff of 3. During the field season the staff is greatly expanded. In 1952, for example, the following were on the exploration payroll: 68 geologists and geological student assistants, 4 surveyors, 4 prospectors, 3 drill foremen, 16 drillers, 10 tractor operators and helpers, 2 mechanics, 13 canoemen, 15 cooks and cookery help, 1 medical officer, 1 radio technician, 2 storekeepers, 4 draftsmen, 3 miscellaneous clerical help, 2 carpenters, 10 laborers, and several other employees who spent part of their time on other projects. Field parties consisted of two 2-man prospecting teams, seven 7-man and three 5-man geological mapping parties, 3 triangulation survey parties, 2 mobile drilling camps, and 6 geologists attached to drill camps. All parties live under canvas and move by canoe, float aircraft, or tractor trains. The company operates 6 small aircraft for servicing and supervising field activities. Aircraft visit parties at least once a week with provisions, including fresh meats and vegetables, mail, and other requirements.

Four supervising geologists circulate among the parties to examine matters of economic interest and to direct the work. The chief geologist makes periodic visits to areas of maximum interest, and the consulting geologist makes two or three visits to the field each season. Field parties are supplied with

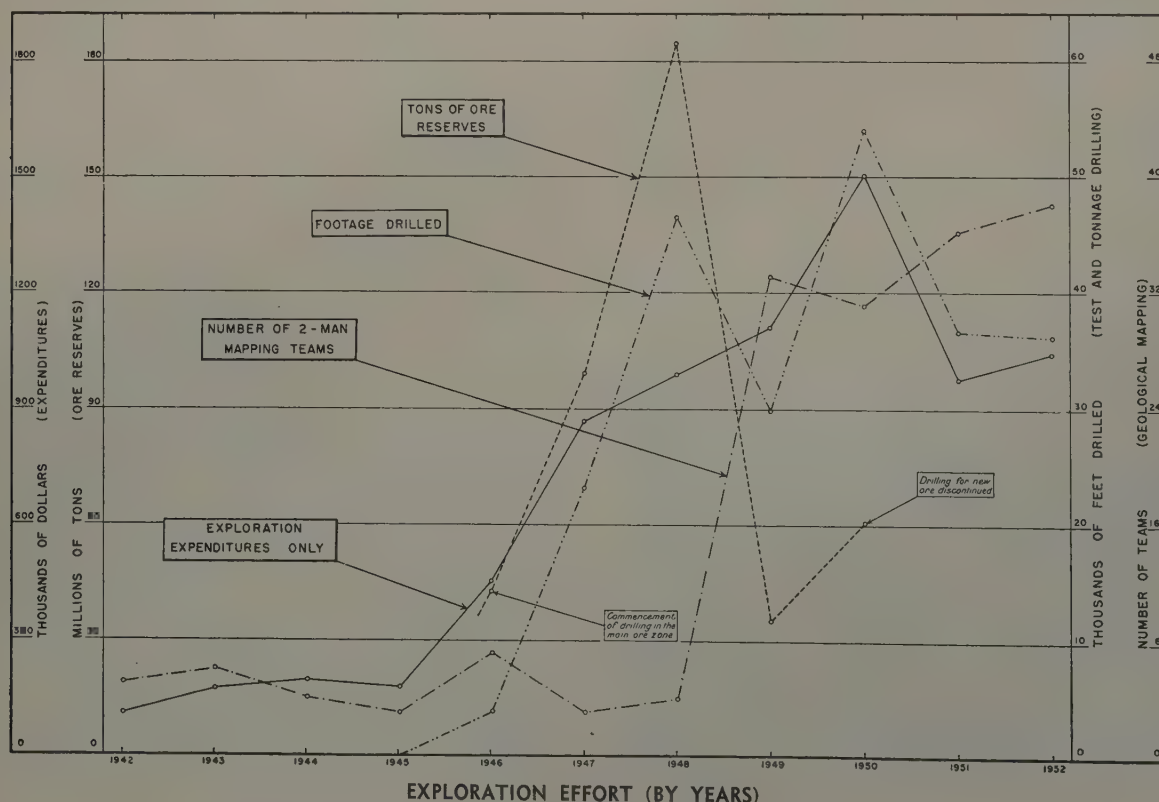


Fig. 7a—Mapping and other exploration effort in recent years.

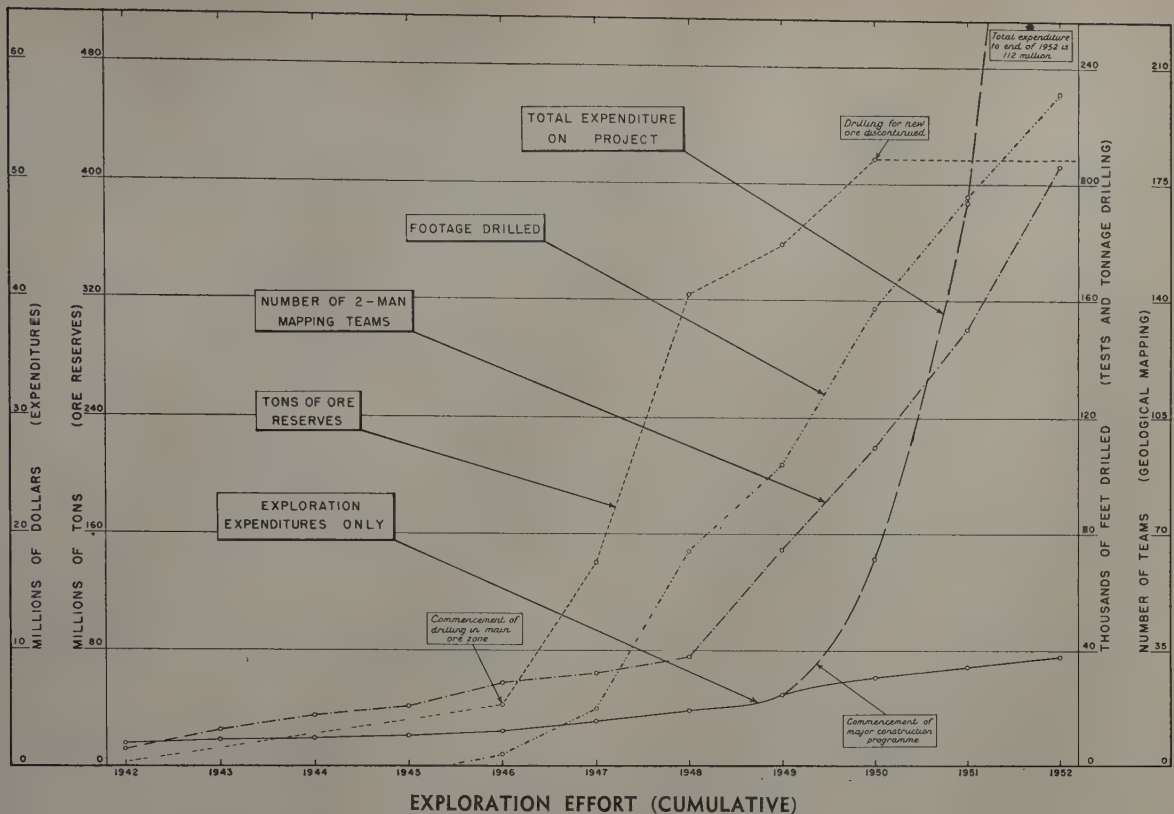


Fig. 7b—The most important economic results of this exploration are given in Table II.

Brunton compasses, dip needles, tachymeters, base maps, aerial photographs, and drafting equipment. Geological information is transferred nightly from aerial photographs and field note books to base maps so that the progress of the work can be readily followed by supervising geologists.

A valuable innovation is a field manual for party chiefs. It consists of some 90 pages and many illustrations covering such subjects as the general geology of the district, including the stratigraphy, structure, and lithology of the rock types; standardized legends and procedures to be used in preparing field notes, maps, and reports; the economic aspects of iron ore geology; and the procedure to be followed when investigating ore showings, as well as an outline of the duties and responsibilities of the party chiefs.

Monthly reports on field operations are prepared by the chief geologist for the consulting geologist and other Montreal and Cleveland personnel. Monthly reports on expenditures and comparisons with budget estimates are prepared by the accounting department.

Exploration and geological policies and programs are under the general supervision of J. K. Gustafson, consulting geologist of the M. A. Hanna Co. Field operations with A. E. Moss in charge are under the operating direction of the manager of mines of Iron Ore Co. of Canada, C. E. McManus.

The cycle of exploration activity in a typical year runs as follows:

1—In December the consulting geologist, after discussing the general scope and cost of proposed exploration for the coming year with the president of Iron Ore Co. of Canada, spends several days at Montreal with the chief geologist and his staff. The previous season's work is reviewed and the results

appraised. The coming season's work is planned in detail, and tentative programs 2 to 3 years hence are drawn up. After discussing the program with the general manager and manager of mines, the consulting geologist prepares a memorandum summarizing work decided on.

2—The chief geologist guided by this memorandum prepares detailed lists of personnel, equipment, and supplies required. An estimate of cash requirements by quarters for the following year is drafted after consultations with the manager of mines, the manager of the subsidiary flying organization, the accounting department, and other company personnel.

3—The consulting geologist reviews and revises the figures with the chief geologist before recommending them to the president and board of directors for formal approval at their annual meeting.

4—In December the chief geologist begins hiring key personnel for the summer operations. Geologists, engineers, and prospectors are usually signed up by February 1st. Other personnel such as drillers, tractor drivers, shovel operators, mechanics, cooks, and clerks are indicated to other departments for hiring. Flying requirements are finalized with the manager of the flying company. Such projects as aerial photography and aerial geophysical work are contracted for.

5—Field work usually begins as soon as the waterways are free from ice around June 15th and continues until the end of September. The operating season once mining starts will be from May until October and is expected to average 5½ to 6 months. Field parties disband, and the permanent staff returns to Montreal early in October.

6—During the first six weeks in Montreal the year-round staff prepares final maps and reports of

Table II. Iron Ore Reserves, in Long Tons, December 1950

	Deposits	Bessemer	Non-Bessemer	Manganiferous	Total
Quebec	25	149,610,000	91,367,000	40,045,000	281,022,000
Labrador	19	78,298,000	45,066,000	13,321,000	136,685,000
Combined Totals	44	227,908,000	136,433,000	53,366,000	417,707,000

Average Grades, Dry Analyses									
	Fe	Mn	P	SiO ₂	CaO	MgO	Al ₂ O ₃	S	Loss
Bessemer	60.73	0.29	0.027	8.71	0.09	0.05	0.59	0.012	2.88
Non-Bessemer	57.53	0.57	0.118	8.07	0.12	0.06	1.47	0.014	6.34
Combined Bess. & Non-Bess.	59.53	0.40	0.061	8.47	0.11	0.06	0.92	0.013	4.18
Manganiferous	50.17	7.64	0.109	7.92	0.10	0.06	1.32	0.012	6.65

field activities. Before December 1st party chiefs employed on a seasonal basis submit final reports and maps covering their areas. Their data are transferred, correlated, and interpreted on master geological plans which are constantly revised. Other winter activities include preparing maps and reports for the Quebec and Newfoundland governments, furnishing information to associated companies, and calculating ore reserves.

Although the attack on new territory has varied the sequence of events in most places has been as follows:

1—Rapid reconnaissance mapping by 2-man parties on a scale of ½ mile to 1 in. along the shorelines of waterways with occasional inland traverses to establish broad geological features.

2—Geological outcrop mapping on a scale of ½ mile to 1 in. along traverses spaced at half-mile intervals by 5 and 7-man parties and cursory examination of most of the iron formation. Two-man teams prospect the most favorable belts concurrently with, immediately preceding, or following this mapping.

Table III. Mapping Completed in Labrador-Quebec Iron Ore District

Scale	Area Mapped, Square Miles
1 in. equals 200 ft	135
1 in. equals 1000 ft	2,610
1 in. equals ½ mile	4,460
Reconnaissance	7,715
Total	14,920

3—Geological outcrop mapping on a scale of 1000 ft to 1 in. of all areas containing bands of iron formation. Traverses are spaced at 500 or 1000-ft intervals depending on local conditions. Practically all outcrops of iron formation are mapped except in areas containing abundant exposures. These areas are usually covered by a second or even a third group of prospectors.

4—Ore deposits and formation containing considerable leaching or enrichment are mapped on 200 ft to 1 in. During this detailed mapping, or immediately following, favorable areas are test-pitted or, where the overburden exceeds 15 ft, test-drilled to delineate orebodies.

5—Preliminary tonnage drilling of orebodies is undertaken and ore reserves calculated.

6—Prior to preparing pit layouts the orebody is mapped on a scale of 100 ft to the inch and continuous surface trenches excavated at right angles to the strike at 200 to 300-ft intervals. The ore is sampled and analysed. Bulk samples are secured

from trenches or from 50 to 100-ft vertical exploratory shafts for testing the physical properties of the ore. Pit outline drilling follows.

Summary of Geological Effort

Figs. 7a and 7b give a quick measure of the geological and other exploration effort in recent years. The most important measurable economic results of this effort are given in Table II, a summary of the grade and tonnage estimates.

Some data not adaptable to graphic presentation may be of interest. The surface area of the 1500 current tracings on file exceeds 10,000 sq ft. This figure does not include field or preliminary maps or the hundreds of plans and sections which have become obsolete. The number of prints made from these originals would undoubtedly exceed 5000. Mapping completed to date is summarized in Table III.

It must be remembered that most of the area mapped on the 200 scale was originally mapped on 1000 scale, most of the area mapped on 1000 scale was originally mapped on the half-mile scale, and so forth. The Dominion government has photographed nearly all of the 25,000 square miles on a scale of ½ mile to the inch, while the Iron Ore Co. and concession companies have had over 3600 square miles photographed on 1000 ft to 1 in. and 500 square miles photographed on other scales. Nearly 40,000 aerial photographic prints are in company files. Fifty-two 2-man prospecting teams have prospected in the area since 1942. Many thousands of test pits have been dug and 231,000 ft of drilling completed. Approximately \$10 million were spent on the exploratory phase of this project.

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- ⁶ M. J. Frarey: *Willbob Lake, Labrador-Quebec*. Geol. Surv. Canada Paper 52-16 (1952).
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Design Development of Crushing Cavities

by H. M. Zoerb

Based on the belief that operating details are a definite contributing factor to major economies, this paper traces the development of crushing cavity design in Symons cone crushers to attain maximum liner utilization. Wear rates are analyzed and compared in this presentation and drawings illustrate succeeding design changes.

IN these times of rising labor and material costs, it has become more and more necessary that attention be paid to some operating details which, in their obscurity, may be the key to major economies. Liner wear in crushing cavities of secondary and tertiary crushers can become an appreciable cost item when the material to be crushed is hard and abrasive. This item of cost not only includes the value of the crushing members, but also more intangible costs such as labor and lost production due to more frequent replacement.

The variables which are encountered in ores and minerals to be reduced; the design of plant and machine application; the sizes, shape, and fineness, characteristics of the crushed product; the moisture; hardness; friability; and abrasiveness of the material to be crushed are all influencing factors which must be taken into consideration in the selection of a crusher, and particularly in the design of crushing cavity and liners to be used in a crusher. Through a research program undertaken in cooperation with many operators of Symons cone crushers a new approach to crusher cavity design was made, resulting in the development of liners for specific operations which showed: 1—maximum utilization, as high as 70 to 80 pct of original weight of metal, and 2—maximum capacity of unit during the greater portion of its life. It has been found that liners so designed for a given operation will show added economies in power consumption, maintenance, and general wear and tear on the crushing unit.

Initial work in the so-called tailoring of crushing cavities was begun on the tertiary or fine crushing units where as a rule reduction ratios were low, varying from 3 to 6. Parallel or sizing zones in the lower portion of the crushing cavity were too long, resulting in a tendency to pack. It was found that very little additional crushing was done in the parallel zone after the initial impact in that zone and that a

relatively small amount of additional crushing was done by attrition, which required very careful feed control. A small amount of over-feeding would result in packing which not only consumed power but caused unnecessary liner wear as well. The illustrations which follow in this discussion will show only contours of crushing cavities, and for purposes of simplification the cavities will be considered only in their closed position.

The first step, therefore, was to reduce the sizing zone to a minimum. This was done by removing the lower portion of the liner as shown in Fig. 1. The result of the change was a saving of 15 to 20 pct in liner cost, less power consumption, with no change in capacity.

This change in design, while an improvement, did not go far enough. As wear took place, the change in the liner was not uniform throughout its entire length, resulting in a restriction of the feed opening and thereby loss of capacity. Furthermore, progressive wear of the liner had the effect of lengthening the parallel zone until finally the entire crushing cavity was all parallel zone, see Fig. 2. It is obvious from the reduced feed opening of the worn liner that the ability of the machine to receive material is lessened considerably. Furthermore, the long parallel zone with its worn, irregular profile did not operate at its highest efficiency.

The first attempt to overcome this difficulty was carried out on a 5½-ft crusher installed in a plant producing roofing granules. The material being crushed was a very hard graywacke and the crusher was closed-circuited with a screen having .232-in. slotted openings. A radical change in contour was developed, as illustrated in Fig. 3. Equal wear lines on both concave and mantle are designated 1, 2, 3, etc. The method of development of this contour is as follows: Since adjustment for wear is vertical, corresponding intersections of wear lines and vertical lines developed concave and mantle contours which maintained equal but lengthening wear surfaces in the parallel zone. The ideal contour, of course, is one in which the length of the parallel zone remains constant, but because of present foundry practice and heat treating characteristics this is impossible.

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Discussion on this paper, TP 3530B, may be sent (2 copies) to AIME before Aug. 31, 1953. Manuscript, Sept. 15, 1952. Los Angeles Meeting, February 1953.

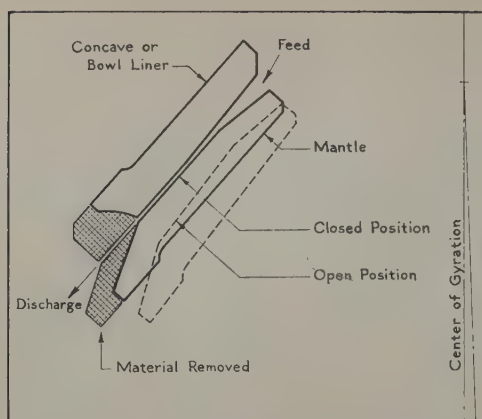


Fig. 1—Section through concave and mantle showing crushing cavity. Sizing zone was reduced to a minimum when lower portion of liner was removed as shown above.

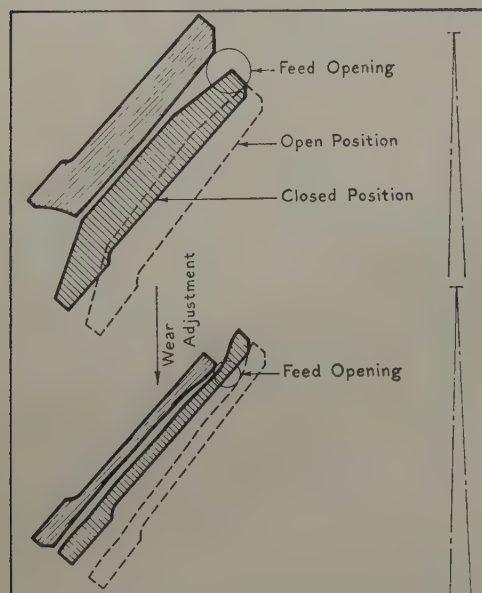


Fig. 2—Lengthening of parallel zone by progressive wear of liner. New liners are shown above, worn liners below.

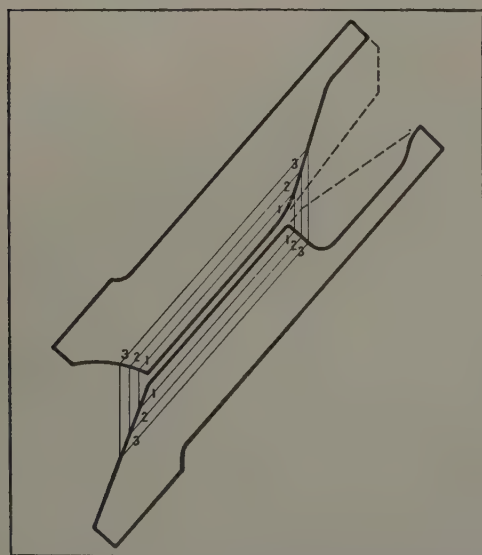


Fig. 3—Equal wear lines on concave and mantle, brought about by new design, are designated by numbered lines. Dotted lines show original contour of upper portion of crushing cavity.

The dotted lines of Fig. 3 show the original contour of the upper portion of the crushing cavity. It may be of interest to note here that the weight of the liners as redesigned was 2345 lb, or 32½ pct lighter than the original long liner illustrated in Fig. 1, and 15½ pct lighter than the conventional short-type liners which have been widely used the past ten years. The abrupt change in section on the mantle caused some concern as to performance, since it was possible that feed would be restricted. Actual performance, however, indicated that these fears were groundless. In fact, there was an advantage in that the rate of feed was controlled into the lower cavity, lessening the possibility of packing.

The result of the change is illustrated in Fig. 4. Solid lines show the worn sections of several contours superimposed upon the original. These contours were obtained by torching a section from the worn members and tracing the contour on a full size section of the original. It is to be noted that a surprising uniformity of wear was obtained. As nearly as can be ascertained, the average capacity increased from 90 to 104½ tons per hr, or 16 pct, and the total average tons crushed per set of liners increased from 71,000 to 84,000 tons, or 18 pct, in spite of the fact that the liners were 15.6 pct lighter. The rate of liner consumption was 0.062 lb per ton crushed as compared to 0.087 lb per ton crushed with the earlier design.

The possibility of this lighter design led to its adoption on other sizes of crushers. On this basis the following conclusions were drawn: 1—Application should be confined to operations involving a relatively small apparent reduction ratio, for example, up to 4. 2—In cases where the feed size was uniformly on the top side, a reduction in capacity would be found. This latter result was due to the restriction introduced by the ledge on the mantle, making the actual feed opening more sensitive to variation in the percentage of voids in the feed.

The next obvious step was to design a crushing cavity for a series of zones, each properly proportioned volumetrically, with proper distribution of metal which would yet result in an economical wear contour. Fig. 5 illustrates a cavity having three dis-

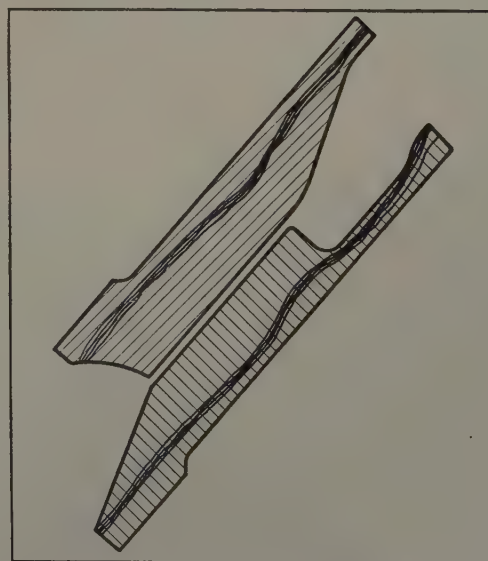


Fig. 4—Solid lines show worn sections of several contours superimposed upon the original. New designing increased average capacity from 90 to 104½ tons per hr.

inct crushing zones, each zone developed according to the method described in connection with Fig. 3. This type of cavity, while still in the tertiary stage of reduction, permitted a greater range of reduction and was less sensitive to feed characteristics in so far as it affected capacity. Furthermore, the volumetric proportion of each succeeding zone required a lower degree of feed control.

An important consideration in the design of a crushing cavity of any gyratory type of crusher is that the angle of nip be held to a minimum. The use of the new design of cavity permitted this. In fact, as can be seen from the illustrations, the angle of nip over the work range of the crushing stroke is almost zero.

Because of the encouraging results obtained with the new type of cavity on tertiary crushers, the basic features were applied in a broader way in the secondary crushing stage. Apparent reduction ratios are generally greater here than in the third stage, although the actual difference is not great. A 12-in. nominal size feed from a primary will seldom have over 10 pct in the 10 to 12-in. size even when passed over a scalping device of reasonable openings. As a general rule, the bulk of the feed size is concentrated in the range above the scalping screen to approximately 60 pct of the top size. A crusher receiving a 12-in. top size feed with a 1-in. setting would then have an apparent reduction ratio of 12, although actually it probably would be 8 or less. It is this greater percentage of material which must receive most attention in the design of cavity and which naturally will be in the lower zones.

By applying the same methods as outlined in Fig. 3 and by departing gradually from the parallel condition above the lower zone, it has been possible to provide large receiving openings in a given machine and yet stay well below maximum nip angles. Furthermore, when the metal thickness of the wearing members is placed where it does the most work, a higher percentage of liner utilization can be obtained, see Fig. 6. A careful study of this problem has enabled us to provide a 7½-in. receiving opening on a 3-ft crusher at 1-in. setting, which was formerly the receiving opening of a 4-ft crusher. The

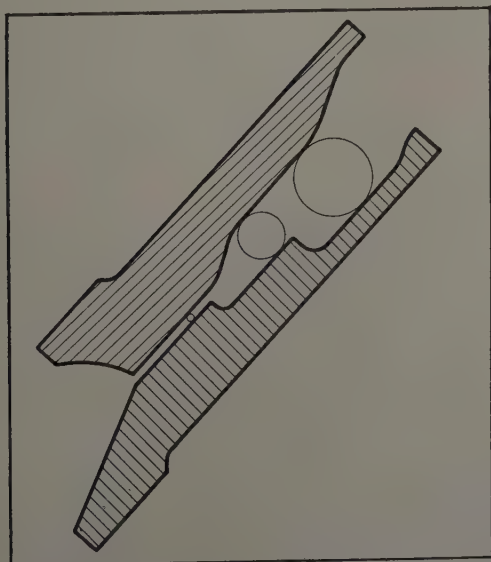


Fig. 5—This cavity, with three crushing zones, permitted greater range of reduction. The volumetric proportion of each succeeding zone required a lower degree of feed control.

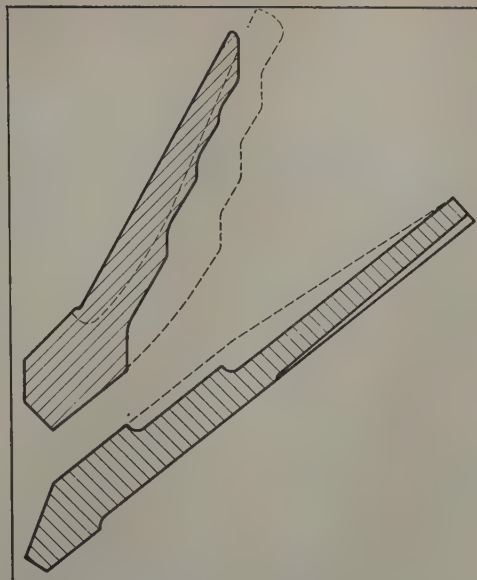


Fig. 6—When metal thickness of wearing members is placed where it does the most work, higher percentage of liner utilization is obtained.

4¼-ft machine can be supplied with a 12-in. opening, which is equivalent to a 5½-ft, and a 5½-ft cavity can be made with a receiving opening of 15 in., equivalent to a 7-ft machine. The 7-ft crusher, formerly limited to 15 in. at 1¼-in. setting, now can have a feed opening of 18 in.

Heretofore, it has frequently been necessary to recommend oversize crushing units to meet feed size limitations, particularly in the case of low-tonnage operations. The redesign of cavities, as outlined here, has permitted a better choice of crushing unit for the duty required, with lower first cost and more economical operation. The use of the so-called controlled feed cavity has enabled a large producer of limestone aggregate to standardize on two-stage crushing plants, producing all required sizes of aggregate except sand.

To date, more than 60 crushing units have been equipped with cavities embodying the features described above. With few exceptions they have been very successful, the exceptions being largely those where a wide range of crushing is done with a single unit. Experience has shown that there is no one type of cavity which will apply to all crushing operations, even though the reduction ratio may be approximately the same. Two crushing units operating in different plants, on different material, effecting the same reduction, may result in entirely different wear contours.

In large-tonnage operations, where the product and tonnage requirements are fairly constant, operating economies can be made by tailoring the crushing cavity to suit that particular condition. If liner wear can be reduced only 0.02 lb per ton crushed, a saving of approximately \$75 per day can be made in a plant handling 10,000 tons. This is not impossible since it amounts to only 15 or 20 pct of many existing wear rates. It is a saving worth going after.

Further investigation remains to be done on this interesting phase of the crushing art. Much has been done in the development of wear-resistant materials and design improvements in grinding mill liners, but the gyratory type of crusher still presents an interesting challenge and until recent years has followed conventional features both as to cavity design and wearing member material.

Testing of Roof-Bolting Systems Installed In Concrete Beams

by Rudolph G. Wuerker

MUCH descriptive matter¹⁻²⁰ has appeared on the subject of suspension roof supports, or roof bolting, as it is more commonly called. The widespread introduction of roof bolting into coal mines and metal mines is truly phenomenal. Mine operators were quick to recognize the advantages of supporting wide openings without hindrance to machine maneuverability and ventilation.

Although suspension roof support has long been installed at St. Joseph Lead Co. mines in southeast Missouri,¹⁰ its application to coal mining presented new problems, such as proper anchorage and bearing for the bolts, bolt diameter, and spacing of bolts. After continuous testing and experimenting at the mines, standard roof-bolting materials were determined.¹⁹

The study reported in this paper is not concerned with such details as bolt diameter, which may be considered already solved in practice. In the tests discussed here, small models patterned on actual bolts were found to function in the same way and as satisfactorily as their prototypes. The aim of these tests was rather to investigate the influence of roof-bolting systems on the stress distribution around mine openings and to study the fracture patterns obtained in actual testing. Little was found about this in the literature, as testing of suspension roof methods and quantitative measurements are only now coming to the fore.

Several suggestions and actual measurements have been made to evaluate critically the functioning of roof bolting systems, single roof bolts, and parts thereof.^{5, 21, 22} Outstanding among them is Bucky's outline of structural model tests.²³ Since none of the suggested testing equipment was available, however, for the experiments discussed below, a different approach was chosen.

The response of a mine roof under stress has often been compared to that of a beam. The slow coming down and bending through of beam or plate-like banks of shale, sandstone, or top coal is a familiar occurrence, extensively cited in the literature.²⁴ It was felt that testing of roof-bolt systems installed in a concrete beam which was loaded in bending would be a fair approximation of the behavior of a mine roof underground.

Another school of thought considers the roof behavior over an underground opening in connection

with the stress distribution all around a circular or rectangular opening. This is more accurate, and leads to the concept of a dome-shaped zone of material destroyed under tensile stress. This is likewise a common sight in unsupported roadways where the continuous fall of roof results in what has been called the natural outline of roof fracture. This theory could not be tested and is treated separately in Appendix B.

It is important to note that according to both assumptions the immediate roof fails in tension; the use of a beam in these tests, therefore, should give information valid for either of the two theories.

With the testing equipment at hand it was possible to load concrete beams 6x1x0.5 ft under two-point loading, giving an equal bending moment over the center part in which the model bolts were installed. A comparison was made of the ultimate loads needed to break plain beams and beams in which roof bolts were installed. Arrangements were made with: 1—plain beams; 2—bolts with plate washers, some with holes drilled at 90° angles and others with holes drilled at 45° angles; 3—bolts with channel irons underneath; 4—bolts in holes filled afterward with cement; and 5—bolts anchored in a stronger stratum.

The foregoing arrangement is made in order of increasing strength, as assumed from the theory of reinforced concrete. Likewise, laminated beams with wooden model bolts and with combinations of the foregoing set-ups were tested. All in all, 21 experiments were made out of the much greater number of combinations possible. There were, too, some trial tests. Enough observations from this limited number were made to interpret the behavior of mine roof, supported by various types of suspension bolts, at fracture.

In present-day concepts, which have been proved by mathematical derivations and stress analyses, any opening driven underground will change the distribution and magnitude of the stresses existing around it. It does not matter whether the stresses become visible, as in rocks whose strength is less than the forces acting upon them, or whether they are invisible, as in the gangways lacking evidence of rock pressure. In this latter case the rocks can withstand changes in stress-distribution.

To consider the mine roof as a beam, there are, with transversal loading, tensile stresses in the lower fiber and compressive stresses in the upper layers above the neutral axis of the beam. Beams of brittle material such as rock and concrete fail exactly as shown in Fig. 1. Nearly all model beams showed the same fracture pattern as that of a tension crack. The influence of support, by roof bolting or conventional

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Discussion on this paper, TP 3554A, may be sent (2 copies) to AIME before Aug. 31, 1953. Manuscript, Feb. 18, 1952. New York Meeting, February 1952.

props, is shown in Fig. 2. Any support has two functions. First, it contributes to the reduction of stresses, as is evidenced by the reduced span of the unsupported rock in Fig. 2. Secondly, it bears the dead-weight of the loosened rock. It is obvious that the stronger the support, the more the stresses will be reduced.

At this point further clarification is needed on the different meanings given to the same strength terms used in many discussions of roof bolting. A distinction should be made, in discussing *strengthening* or *reinforcing* a mine roof, as to whether the increase is from a low collective strength to the highest value possible for this material, that is, from a laminated to a more competent beam, or whether it is a further increase from this point on, from a plain but competent beam to a still higher value, much as a concrete beam is reinforced by the installation of horizontal rods and vertical stirrups, see Fig. 3, lower half. The first case holds for all discussions of the benefit of using bolts on *lamination* of strata, and in the discussion in Appendix A it is shown that no strengthening of a laminated beam over and above the strength of the competent beam is possible. But this was not the aim of the tests, although two beams, No. 21 and 22, were cast in two halves each 6-in. high within an interval of two days. As the results were inconclusive this was not followed up.

This distinction in the two different types of beams is reflected by using the term *bearing capacity* for laminated beams, in analogy to soil mechanics, and *strength* for competent beams to which the laws of elasticity can be applied.

The purpose of the experiments was explicitly to study the second step in increase, to discover whether or not the bolts would increase strength in the same way horizontal bars and stirrups increase the strength of the common reinforced concrete beam of the structural engineer.

Testing Procedure

In the tests, concrete beams 72 in. long, 12 in. high, and 6 in. deep were supported over a 64-in. span. A two-point loading was applied by a girder, whose loading points, P , were 32 in. apart. The girder in turn was pushed down by the head of the testing machine. Thus a constant bending moment throughout the center section was given. In this case, see Figs. 3 and 4, the maximum bending moment is $P a$, the maximum deflection

$$\Delta \max = \frac{P a (3l^2 - 4a^2)}{24 E I} \quad [1]$$

wherein

$\Delta \max$ = maximum deflection, in.

P = load, lb (Note that the load exerted by the plunger of the testing machine equals $2P$.)

l = total span of beam, in.

a = distance between support and load, in.

E = modulus of elasticity in bending, psi

I = moment of inertia, in.⁴

From the moment diagram, Fig. 4, it can be seen that there is an equally distributed load between points BC, where the loads act and between which the model roof bolts were installed. Within this 32-in. part of the beam four holes were drilled 9 1/3 in. apart, some at an angle of 90°, some at 45°, so that it was possible to test all roof-bolting systems either of steel or wood with varying anchorage or

with plates or ties under the tension side of the beam. The plain beams were broken first and the relative increase in strength was sought, as expressed in higher loads on the plunger of the testing machine, and as caused by the roof-bolting systems.

Preparation of Concrete: The beams were cast with concrete of a ratio, by weight, of 3.5 parts sand to 1 part cement to 0.53 parts water, the last value containing 1.5 pct free moisture of the wet sand. This ratio was chosen as being close to the tensile strength of roof shale previously determined by extensive testing. The maximum size of the sand, which met all the standard ASTM specifications, was 1/4-in.

Through the cooperation of the management of the Murdock Mine of the Bell and Zoller Coal and Mining Co., Murdock, Ill., samples of shale from roof fall had been obtained that could be taken as typical for a mine roof over Illinois seam No. 7. Tensile strength of this shale was determined by a method similar to the ASTM standard C-190-44 for hydraulic cement mortar and by means of Riehle cement briquet testing machine. Average value obtained was 770 psi. The specimens, however, were cut from sound shale banks that might not be representative for the collective strength of this particular shale roof, which is full of bedding planes and shear cracks.

But a concrete of such high tensile strength is not common. The above-mentioned mixture is the result of a great many tests to approach the value of the tensile strength of the roof shale. Only a few specimens reached an 800-psi tensile strength at an age of 2 to 3 months. Three tensile specimens were prepared separately on the day of casting of each beam, following ASTM Standard C 190, and tested the same day in the Riehle briquet tensile test machine. Likewise, separate 2x4 in. cylinders for compression tests were set aside the day of casting of a beam, and compressive strength determined at the day of breaking the beam.

All beams, tensile and compression specimens, were moist-stored at a temperature of 70°F for 28 days, and then taken out for drilling holes, installing the bolts, and testing. The holes were 10 in. deep and were drilled with a 3/8-in. drill. In simulating underground conditions, drilling was favored over precasting, since drilling a hole may weaken the rock and act as a stress-concentrator.

Preparation of Steel Roof Bolts: The model steel bolts, shown in Fig. 5 with all accessories, were of 5/16-in. cold-rolled, round steel with a tensile strength of 60,000 psi at the proportional limit. They were 1/16-in. less in diam than the hole. The torque applied to the bolts was 20 ft-lb. Afterwards the individual bolts at each broken beam were tested with a torque-wrench. It was found that the wedge anchorage was sufficient and that the bolts were sheared off in or just above the threads at torques of 20 to 25 ft-lb.

Preparation of Wooden Roof Bolts: The pertinent data concerning wooden model bolts, washers, and plates, made of red oak, is given in Fig. 6.

Twelve specimens of the oak wood, used for the wooden roof bolts, were tested in tension. The extreme and average values obtained are shown in Table I.

Single and double shear strength of the red oak was determined by means of a conventional fixture for testing metals in shear. Load was applied normal and parallel to growth rings. This distinction need

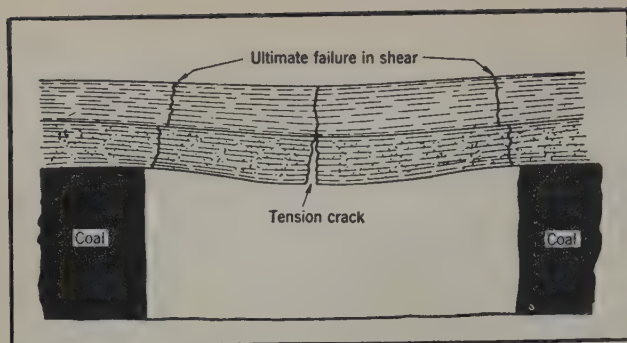


Fig. 1—Action of immediate roof preceding failure (from U. S. Bur. Mines Handbook: *Questions and Answers on Roof Support in Bituminous-Coal*).

not be made with roof bolts underground, however, and the following statistics are therefore listed: lowest value, 4700 psi; highest value, 5330 psi; average value, 5045 psi. It should be mentioned that double shear gave about twice the above values, as was to be expected.

Test Results of the Material: In spite of all precautions taken, very few definite strength and elastic values of the concrete were obtained. Concrete, however, behaves much like sedimentary rocks of mine roof. Concrete is, according to data available, weaker than shales, schists, and sandstones, but as it can be formed to almost any shape it is an ideal material for testing of sedimentary rocks. Some strength and elastic properties of concrete increase with age, as can be seen in Table II, columns 4 and 5 for compressive strength and columns 4 and 6 for tensile strength. In the case of the tensile property, an increase in tensile strength with age from about 400 psi for specimens 23 and 24, of 22 and 25 days age respectively, to 550 to 600 psi as average for specimens older than 28 days is within expectations. The compressive strength of the separate specimens, which is needed for computing the modulus of elasticity, showed a slight increase with age. Abram's

$$\text{formula } S_{\text{compr.}} = \frac{14,000}{7 \left(\frac{w}{c} \right)}$$

wherein $\frac{w}{c}$ is the water-cement ratio by volume, seems to coincide with the arithmetic mean of all

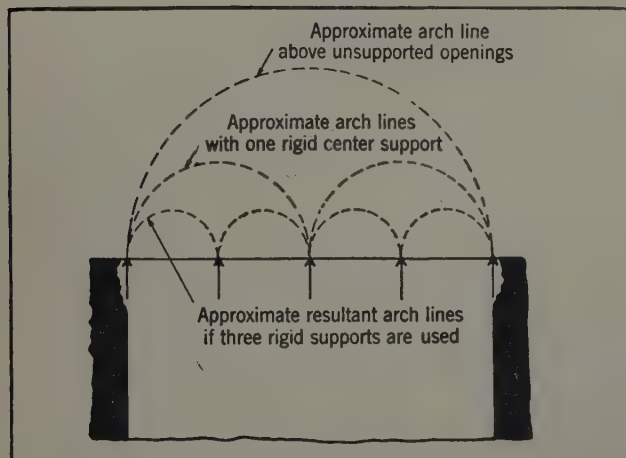


Fig. 2—Approximate arch lines above opening (from U. S. Bur. Mines Handbook: *Questions and Answers on Roof Support in Bituminous-Coal*).

compressive strength values plotted. Its numerical value in the present case was 6230 psi. However, in bending no increase of strength with age is assumed after 28 days, rather, a slight decrease; thus age need not be considered a variable in testing work.

Test Results of Roof Bolting Systems: Whereas four repetition tests could be made with plain beams, this number of tests for each possible arrangement would have been too costly and time-consuming. Duplicate tests were run only for the most common arrangements: 90° holes with plate washer under, 90° holes with channel iron under, and 90° holes with plate washer but with anchorage in a stronger stratum. This last arrangement was effected by opening up the beam from top, putting a ring washer under the wedge and closing the hole in the beam again with a quick-setting cement or a mixture of cement and plaster of paris.

All other tests were made with a single specimen. They might or might not be representative for that special bolt installation, giving a value either too high or too low, similar to the wide spread of strength properties of concrete in plain beams.

Table I. Results of Parallel-to-Grain Tension Tests of Red Oak

Item	Moisture Content, Pct	Sp Gr	Modulus of Elasticity, Psi	Ultimate Tensile Strength, Psi
Lowest value	8.5	0.72	2,258,000	16,230
Highest value	8.2	0.69	2,459,000	30,890
Average value	8.5	0.70	2,347,000	22,990

The complete test results are listed in Table II. The most important data is found in column 3, Description of Beams, and column 11, Type of Failure. The behavior of a special roof-bolting system at or after the fracture of the beam can here be seen quickly. The ultimate load exerted by the plunger of the testing machine is listed in column 10. This value is equivalent to $2P$, P being used as in Figs. 3 and 4.

The strength data of separate test specimens is likewise given. The average compressive strength and the modulus of elasticity derived therefrom are found in columns 5 and 8 respectively. Average tensile strength of separate specimens is listed in column 6, and data pertaining to the beams in column 7, Modulus of Rupture, and column 9, Modulus of Elasticity in Bending. An interesting

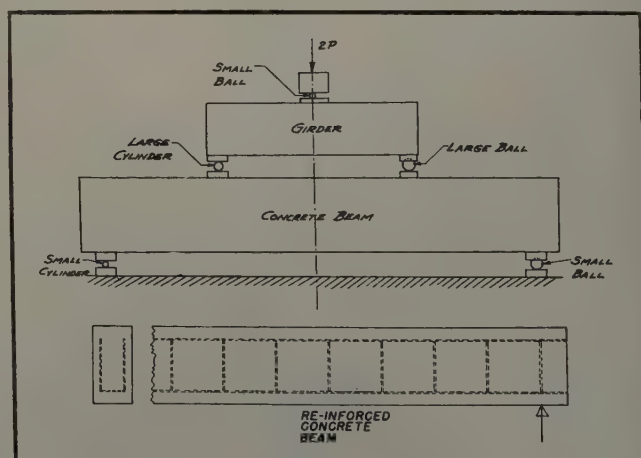


Fig. 3 (Upper)—Arrangement for testing roof bolting systems in concrete beams, and (lower)—reinforced concrete beam.

Table II. Tests of Roof-Bolting Systems Installed in Concrete Beams

1	2	3	4	5	6	7	8	9	10	11	12
Exp. No.	Beam No.	Description of Beams	Age on Date of Test (Days)	Avg. Compr. Strength on Date of Test (Psi)	Avg. Tensile Strength on Date of Test (Psi)	Modulus of Rupture from Bending (Psi)	Modulus of Elasticity in Compression (10 ⁶ Psi)	Modulus of Elasticity in Tension (10 ⁶ Psi)	Load at Failure, Lb	Type of Failure	Bolt Arrangement and Location of Failure
1	13	Plain	36	7775	567	236	2.045	3.16	4250	Tension	
2	14	Plain	36	8500	667	139	1.530	1.08	2500	Tension	
3	23	Plain	25	5493	450	278	1.842	1.290	5000	Tension	
4	24	Plain	22	4803	366	263	0.928	1.59	4750	Tension	
5	10	Holes, but no bolts	40	6910	478	200	1.285	0.980	3600	Between holes 2 and 3	
6	3	Steel bolts, 5/16-in.	50	4800	579.2	264	1.275	1.37	4750	Through hole 3	
7	4	Steel bolts, 5/16-in.	78	5560	568	292	1.000	2.28	5250	Through hole 2	
8	1	Steel bolts, 5/16-in.	64	4500	538	250	1.010	0.892	4500	Through hole 2	
9	5	Steel bolts, 5/16-in.	43	3940	748.3	306	1.720	1.630	5500	Shear outside	
10	11	Steel bolts, 5/16-in.	64	6350	750	322	1.290	2.180	5800	Shear outside	
11	7	Steel bolts, 5/16-in.	78	7500	800	300	1.194	1.570	5400	Tension, through hole 1	
12	12	Steel bolts, 5/16-in.	64	7950	770	356	1.389	2.52	6400	Between 1 and 2	
13	6	Steel bolts, 5/16-in.	92	6950	732	333	1.171	1.45	6000	Shear through 1 at top	
14	19	Steel bolts, 5/16-in., inserted on casting	43	8040	533	272	1.310	1.52	4900	Tension between 3 and 4	
15	20	Steel bolts, 5/16-in., inserted on casting	43	7400	568	372	1.150	1.210	6700	Through 2	
16	21	Steel bolts, 5/16-in., laminated beam	46	5696	488	221	1.272	1.650	4000	Through 3	
17	22	Steel bolts, 5/16-in., laminated beam	46	6015	505	278	1.613	1.360	5000	Through 1	
18	9	Wooden bolts, 3/8-in.	89	6896	266	400	2.000	1.770	7200	Between 1 and 2	
19	15	Wooden bolts, 3/8-in.	32	4160	653	333	0.722	1.54	6000	Between 2 and 3	
20	18	Wooden bolts, 3/8-in.	53	8770	588	278	2.545	1.14	5000	Between 1 and 2	
21	16	Wooden bolts, 3/8-in.	46	5600	700	333	1.120	2.18	6000	Tension through hole 3	

* Fracture outside bolted area.
** Beam still held together.

observation as to the validity of the laws of elasticity, on which the experiments were based, can be made here. For plain beams, 1 to 4, the values of columns 6 and 7, Tensile Strength and Modulus of Rupture, and the values of columns 8 and 9, Elasticity in Compression and Bending Test, should show better agreement. The same spread of experimental data could be observed in the breaking load of the plain beams in column 10. In testing brittle materials such as concrete or rock the experimenter must be satisfied with the fact that strength data are rarely definite but will spread over a certain band. For this reason no other measured or computed data is listed, although many more are available. The experiment was, except for the nature of the material, well under control. A further improvement, like installing strain gages on the bolts, would be worth following up.

Interpretation of the Test Results. Increase in Strength: In an evaluation of test results, see Fig. 7, breaking load is plotted as a function of increase in reinforcement. As the arrangement of increasing effectiveness of the various roof-bolting measures is a somewhat arbitrary procedure, the five steps previously mentioned were followed, namely: 1—plain beams, 2—roof bolts with plates only, 3—with headers or channel irons under, 4—with better bond between steel and concrete, and 5—better anchorage, which was brought about in the tests by the placing of a washer under the inner wedge in imitation of actual roof bolts anchored in a stratum stronger than the immediate roof. As some of the beams contained overlapping properties, this way of plotting could hardly be carried any further, but the contribution to increasing strength is clearly evident in Fig. 7.

Failure Pattern: More convincing conclusions could be drawn from the study of the fracture patterns, shown by symbols in column 12 of Table II. Out of 17 tests with bolts, six beams failed through 90° holes, six with tension cracks between the 90° holes. Two beams cracked through, but not along

45° holes, and three broke outside the bolt system. The following results stand out.

1—All the beams which had a channel iron connecting the bolts at the bottom of the beam, that is, No. 9, 10, and 13, failed outside the bolts, and as a glance at Fig. 7 shows, had relative high strength values also. They were the only ones that did not break with a tension crack, but in a pattern that approached shear failure. The portion of the beam over the channel iron was apparently strengthened to such an extent that the fracture took place in the unsupported part. Another beneficial influence of the headers under the bolts could be observed in the case of experiments 20 and 21. Here the beam failed in tension within the bolting system, but the broken beam was still held up by the header. Applied to mining this would mean that fall of roof would be avoided or delayed until additional support could be put up.

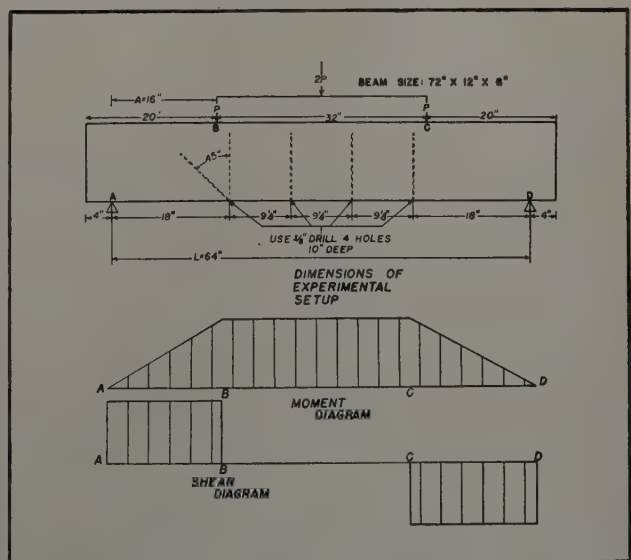


Fig. 4 (Upper)—Dimensions of experimental set-up, and (lower)—shear and moment diagram.

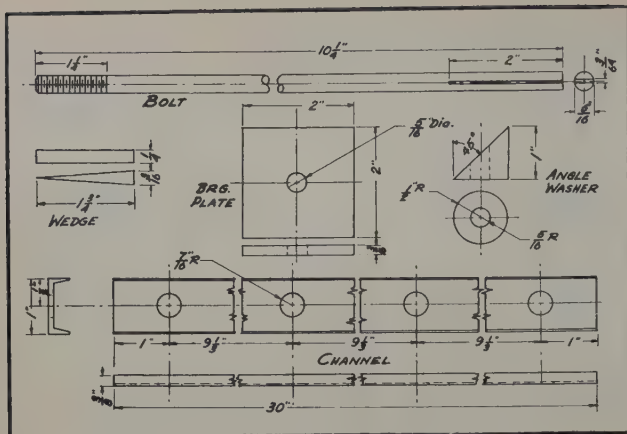


Fig. 5—Model steel roof bolt with accessories.

2—This same effect of delaying roof fall was noted in cases where the beam fractured through a 45° hole, but at a different angle, see experiments 15 and 21. The broken beams were held together by the bolts in a way similar to the beams with roof bolts and headers. Roof fall in this case would also have been delayed.

3—The behavior of wooden roof bolts in the tests corresponded in all respects to that of steel bolts. On Fig. 7 the wooden bolts, beam numbers 9, 15, and 16, all can be found in the upper range of breaking loads. The failure behavior showed that the wooden bolts and boards would have sufficient strength to hold broken roof portions about to fall.

To summarize, on the assumption that suspension support aims mainly at reducing the extension of the zone of destroyed material, it can do so only if it can strengthen the immediate roof.

Influence of Vertical and Lateral Reinforcement: It is evident that the greatest disadvantage of bolting is the gap formed between the wall of the drilled hole and the roof bolt. If the roof in flexure bends through, the material will move into the space left between bolt and circumference and in doing so has already lost its cohesion. This explains why 50 pct of the tested bolts with 90° hole and plate under cracked through the hole. The holes acted as stress-concentrators.

Although the strengthening effect brought about by the bolts in the other 50 pct of the beams was sufficient to overcome this detrimental effect, it was not sufficient to strengthen the beam laterally. The beams cracked between the holes. These shortcomings can be overcome by the following measures, listed in their rank of efficiency: 1—roof bolt having the full diameter of the hole, as in the case of wooden bolts. 2—high friction between bolt and rock, as with the wooden bolt, or the steel bolt, if it is the deformed reinforcing round bar type. 3—perfect bond between bolt and wall of hole. This is the mine operator's intention when he injects cement in the gap between bolt and rock. The technical and economic limitations of this method are obvious.

The above procedures might bring about vertical reinforcement, but their effect in a horizontal direction seems insufficient. This was brought out by the experiments in which one half of the bolted beams failed in between the holes. This can be observed underground on certain shale roofs, see Fig. 8. In soft shale roof, fortunately not common, the rock falls out around the bolt supported by the plate only and tapered upward. This failure pattern is certainly a good example of the stress distribution

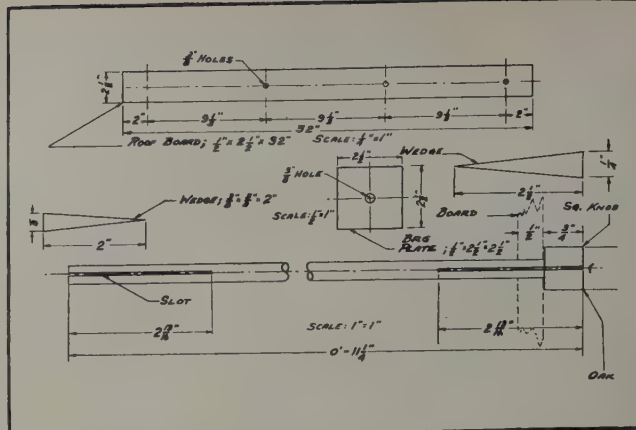


Fig. 6—Model wooden roof bolt with wedge, board, and plate.

around a vertical bolt. The similarity with the theoretical arch lines above a supported mine opening in Fig. 2 is indeed striking. The same phenomenon has been observed by Cross and Price²² in other mining areas.

A further conclusion can be drawn as to the advantage of lateral reinforcement. Only in the cases where channel irons and headers were installed was the lateral strengthening apparently sufficient to bring about higher breaking loads and to shift the plane of final fracture outside the stressed zone. A logical step from here on would be to install headers in the longitudinal axis of the roadway. At the face they would cover the unprotected area in much the way forepoling is used. A second step would be to make a still more complete skin by putting up a wire of ordinary fencing wire from bolt to bolt. This would certainly keep any loose rock from falling. Cross and Price,²² carrying out their observations underground, arrived independently at the same suggestion.

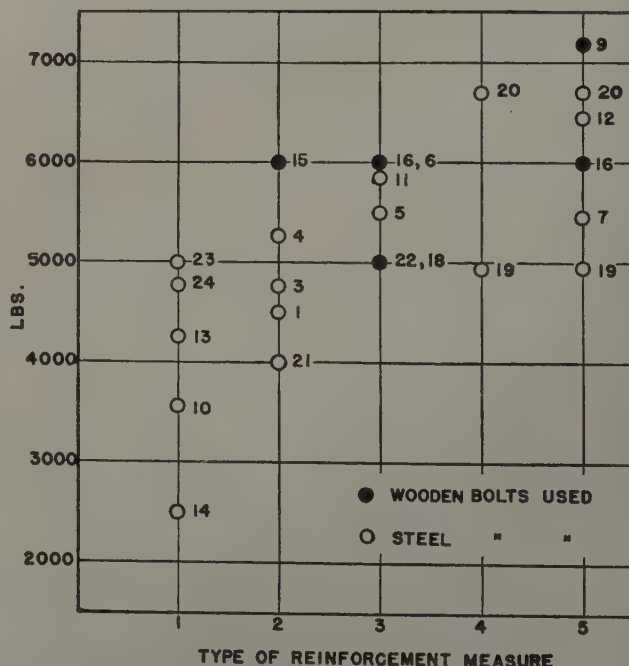


Fig. 7—Ultimate breaking load of concrete beams and roof bolts tested vs reinforcement measures, which are: 1—plain beams; 2—bolts not cemented in place, plates only; 3—channel iron, or board in the case of wooden bolts, on underside of beam; 4—bolts inserted in casting; 5—better anchorage. Numbers 1-24 refer to beam numbers given in Table II, column 2.

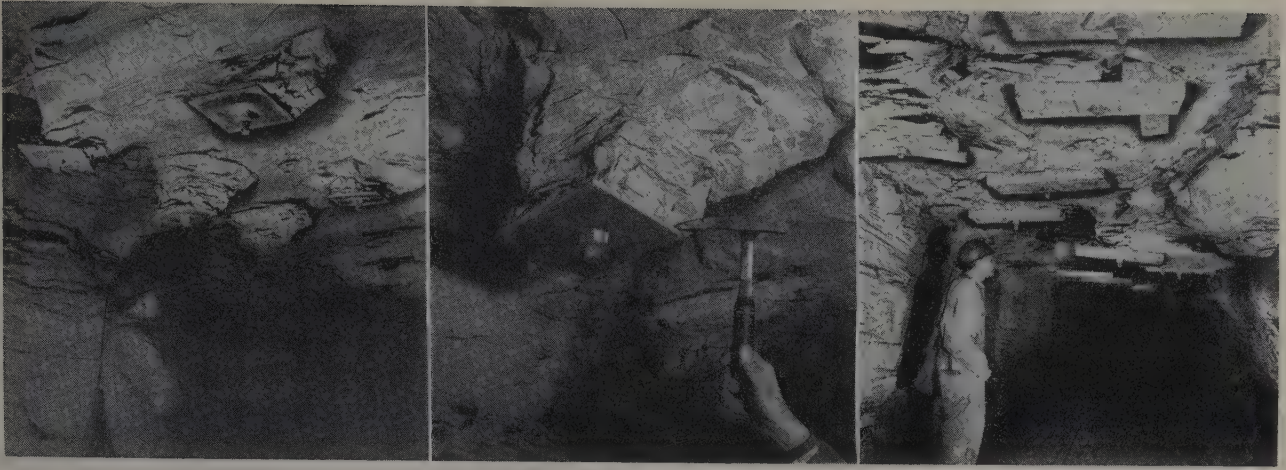


Fig. 8—Destruction of soft shale roof around roof bolts. Left, shale falls out around base plate and leaves a cone of rock supported by base plate. Center, enlarged view of base plate. Right, wooden blocks over base plate.

Influence of 45° Hole on Fracture: Because of the tendency of concrete beams to fail with a tensile crack and at a 90° angle, the beams with a 45° hole, see experiments 15 and 21, showed a certain advantage over the 90° hole in the sense that after failure the beam was still held up by the bolt. It could be reasoned that the holes could have been drilled in such a way that their projection would have fallen over the whole length of the beam, so that any crack would have to intersect a bolt. This is important when bolts are installed at a 90° angle and are held by plates.

Other roof rocks will certainly have a different fracture pattern; they will probably tend toward shear failure. If there is a prevalent tendency to break at a certain angle, it should be possible to set the holes and bolts so that they will intersect the preferred fracture planes. Another advantage of the 45° hole is that it allows the installation of longer bolts than the 90° hole, a fact which becomes of importance in the case of low seams.

Wood as Bolting Material: From the testing results and the foregoing considerations it is evident that wood meets most requirements outlined for an ideal roof bolt. Provided that no detrimental atmospheric and water conditions exist, the relative high strength of wood exerted by a much larger cross-sectional area comes into action. Using the value of 16,230 psi as tensile stress for wood and 30,000 psi for steel, a 1-in. steel bolt would sustain a load of 25,400 lb at the proportional limit, and 50,800 lb at the breaking point, in contrast to 88,000 lb taken up by the wood bolt at the proportional limit. The slotted ends of a wooden bolt, however, will probably fail before this value is reached.

A similar comparison applied to the shear behavior of the two materials gives a higher ultimate shear load to the steel bolt, in spite of its smaller cross-sectional area, by reason of the fact that the shear-strength of steel is about ten times that of red oak.

As wooden roof bolts can fill the hole drilled in the roof, they offer a further advantage. No initial movement of the rock into the hole seems to be possible, and a certain amount of friction can be assumed that might substitute for the bond between concrete and reinforcement postulated in the foregoing discussion.

Other Functions of Support: It should be borne in mind that stress or load is not the only factor in-

fluencing the strength of rock. Generally, the state of a material like rock can be explained as a function of the following five factors:²⁵ 1—load, in this paper only discussed; 2—stress history; 3—chemical and physical environment; 4—time; 5—temperature.

With sedimentary rocks, items 3 and 4 in particular often influence the behavior of roof and walls more than do the stresses set up in them. The exclusion of humid mine air has been described rightly in the literature as one of the most beneficial effects of roof bolting on the conservation of mine roof.

The influence of time on loading, often alluded to as *creep*, has not been stressed enough. While rock creeps at ordinary temperature, steel does not. Roof bolts will therefore stop the slow coming down of the roof so common in mining. This is caused by the fact that rocks will deform under loads much lower than the ultimate load, if the load is applied only over a period of days or weeks.²⁶⁻³¹

Appendix A

Rewriting Eq. 1

$$\text{Maximum moment } Pa = \frac{\Delta_{max} \cdot 24 \cdot E \cdot I}{(3l^2 - 4a^2)}$$

$$P = \frac{1}{a} \frac{\Delta_{max} \cdot 24 \cdot E \cdot I}{(3l^2 - 4a^2)}$$

It becomes evident that the load required to break a beam increases if Δ_{max} , E , and I increase. The deflection Δ_{max} is not important here, and E is supposedly

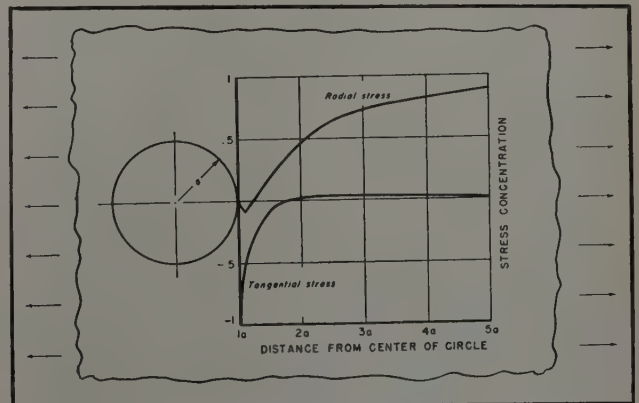


Fig. 9—An areal distribution of stress along vertical axis of symmetry for circular hole in infinite plate. Unidirectional stress field. (After Duval, U. S. Bur. Mines R. I. 4192).

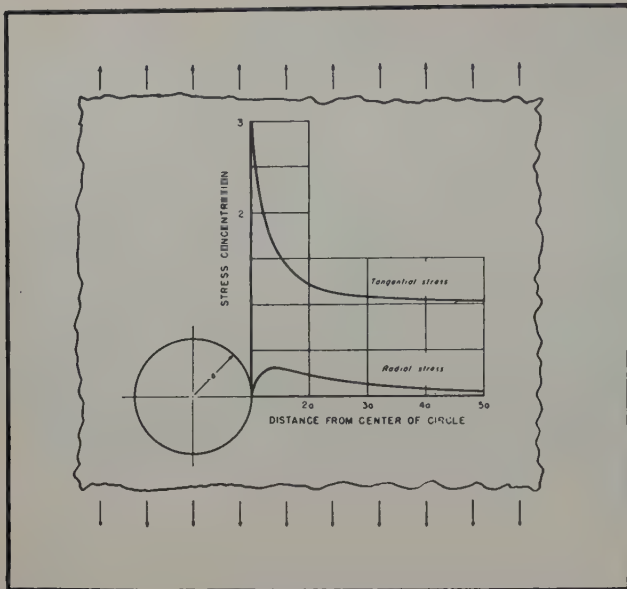


Fig. 10—An areal distribution of stress along horizontal axis of symmetry for circular hole in infinite plate. Unidirectional stress field. (After Duval, U. S. Bur. Mines R. I. 4192).

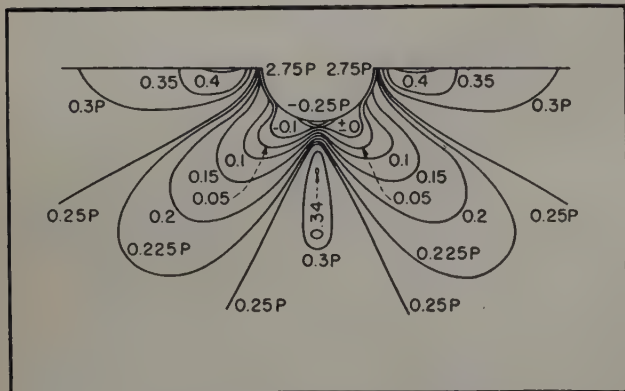


Fig. 11—Lines of equal stresses around lower half of circular opening. Positive values are compression stresses; negative values are tension. (After Wuerker, *Engineering and Mining Journal*, June 1949, p. 61).



Fig. 12—Zone of destroyed and slaty material in floor of drift, similar to the stress distribution in Fig. 11. (After Wuerker, *Engineering and Mining Journal*, June 1949, p. 61).

a constant for any material. It could be argued that in suspension support E_r of rock plus E_s of the steel bolts would combine to a higher value than E_r . This would be analogous to reinforced concrete, where E of the reinforced member is higher than E of the plain concrete. But in this latter case, steel and cement make a perfect bond. This does not take place with conventional roof bolts, as is discussed under *Influence of Vertical and Lateral Reinforcement*. This can be seen, too, by comparing the modulus of elasticity in compression, see column 8, Table II, with the modulus of elasticity in bending, column 9, Table II. If there were such an increase in strength properties, as is obtained by the steel bars in reinforced concrete, the modulus of elasticity in bending of the bolted beams should be much higher than the one in compression of the plain concrete. Whatever increases can be noted, they are not higher than the variation in properties inherent in a material like concrete.

The only quantity left that might influence P , then, is I .

$$I = \frac{bh^3}{12}$$

where b = width of beam or lamina and h = height of beam or lamina.

If a beam of the height h_b is made up of n laminae having the height h_i which is always a fraction of h_b , the best bond or lamination of all the laminae will never reach a height Σh_i greater than h_b .

Thus a strengthening of a laminated beam over and above the strength of a competent beam is impossible.

Appendix B

In the experiment, roof bolting systems were treated as reinforcements of a beam under transverse load. There is, however, a second school of thought that considers the changes in magnitude and distribution of stresses all around an underground opening. This is, no doubt, the more accurate view. The nature of these changes has been investigated by Love,³² Mindlin,³³ Schoemaker,³⁴ and Wuerker.³⁵ Photoelastic tests³⁶ have shown very good agreement with the theoretical assumptions.

Figs. 9 and 10, as given by Duval,³⁷ explain best the point in view. Here the stress concentration, that is, the stress at any point divided by the average applied stress, around a circular opening in a unidirectional stress field is plotted against distance from center of hole. Fig. 9 shows the stress distribution along the vertical axis and Fig. 10 the stress distribution along the horizontal axis. In this latter case, the tangential stress produced on the boundary at the ends of the vertical axis is equal in magnitude but of opposite sign to the applied stress. Thus if the applied stress is a compression, this stress is a tension. Failure of rock occurs, therefore, in the roof and floor rather than in the side walls, as the tensile strength of most rock is only a very small fraction of the compressive strength.

This is the explanation for the formation of the well-known *dome* over openings, which is a zone of rock so destroyed by tensile stresses that all coherence is lost and the roof starts falling. If lines are plotted of equal stress around the circular opening, a stress distribution as shown in Fig. 11 results, giving lines of equal principal stresses with the secondary stresses suppressed. (A symmetrical identical pattern exists over the upper half of the roadway.) Negative tensile values can be seen at the top and bottom of the opening.



Fig. 13—Formation of dome in roof of roadway. (After Wuerker, *Engineering and Mining Journal*, June 1949, p. 60).

Both Figs. 12 and 13 are photographs which were taken underground. They show the formation of domes in the roof and floor of a country rock. Original bedding planes can be seen from the lighter and darker bands in Fig. 14 and are shown dipping about 10° toward the right. The conformity with the stress-analytical picture is indeed convincing.

Roof bolting, as practiced in our coal mines, does not concern stress all around the openings. Should need for this arise, probably in very deep mines, it would be sufficient to strengthen the immediate roof and sidewalls to such an extent that the stress concentration of $3P$, see Fig. 11, is taken care of. This takes place in a zone with a radius of $2a$, distance a being equal to the radius in the case of circular opening. In other words, roof bolts should have at least the length a in this example.

Acknowledgments

This testing work has been carried on by the Department of Mining and Metallurgical Engineering with the cooperation of the Department of Theoretical and Applied Mechanics of the University of Illinois, as part of the work of the Engineering Experiment Station of this university. Professor H. L. Walker, head of the first-named department, has given all support possible to the project. The Department of Theoretical and Applied Mechanics took charge of preparing tensile and compression samples of concrete beams and of preparing special fixtures for the test. The great help given in this phase by Assistant Professor C. E. Kessler, in charge of the Concrete Laboratory, and by his staff and shop personnel is gratefully acknowledged.

The Department of Mining and Metallurgical Engineering prepared the models of the various roof bolts, installed them, and ran the tests. Credit is due Messrs. N. Szabo and, especially, S. J. Jachim, who did most of the preparing and executing of the tests. Former students B. W. Gilbert, A. Risi, and J. W. Snarr helped during the actual tests and did most of the recording, plotting, and computing of experimental data. In this part of the work the shop personnel of the Department of Mining and Metallurgical Engineering under H. W. Kruse, Senior Laboratory Mechanic, helped readily and most efficiently so that the experiments could be carried out on the rigid schedule prescribed by the change in strength properties of concrete with age.

Supplementary work was carried out by G. M.

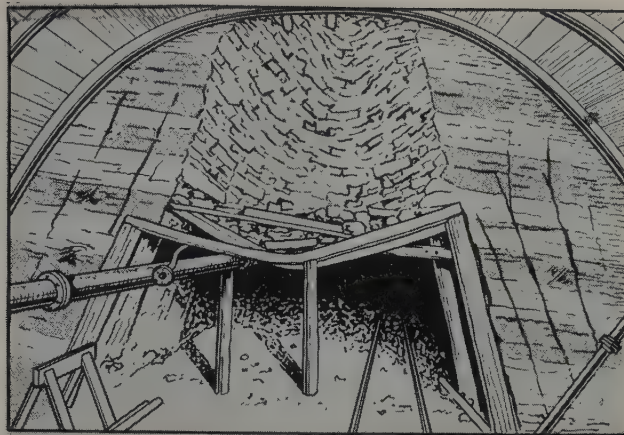


Fig. 14—Reproduction of Fig. 13 to show zone of crushed rock in the dome over the square set. (After Wuerker, *Engineering and Mining Journal*, June 1949, p. 60.)

Hunt, Director, Forest Products Laboratory, U. S. Forest Service, Madison, Wis., and his staff in determining some strength values of the wood used in wooden roof bolts.

The manuscript was read by Assistant Professor C. E. Kessler, Department of Theoretical and Applied Mechanics, by Associate Professor G. B. Clark, Department of Mining and Metallurgical Engineering, and by T. E. Coleman, Jr., Assistant in English and in Metallurgical Engineering; their various suggestions are gratefully acknowledged.

The author was in charge of general planning and supervision, as well as the writing of this report.

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Reducing the Moisture Content and Large Moisture Variations in Russellton Washed Coal

by Orville R. Lyons

THE Russellton preparation plant of Republic Steel Corp., located north of Pittsburgh, Pa., prepares 3½x0-in. Thick-Freeport coal by means of a 13-ft 6-in. diam Chance cone and 16 No. 7 Deister tables. The plant feed ranges between 370 and 410 tons per hr, of which approximately 31 pct is ¼x0-in. size, and the plant reject ranges between 14 and 16 pct of the feed.

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Discussion on this paper, TP 3506F, may be sent (2 copies) to AIME before Aug. 31, 1953. Manuscript, Aug. 13, 1952. Los Angeles Meeting, February 1953.

The 3½x¼-in. washed coal, the cone product, is dewatered and desanded by a double-deck Parrish-type shaking screen 46 ft long and 7 ft wide. The 1¼x¼-in. portion is further dewatered by two 4x12-ft single-deck vibrating screens. The ¼x0-in. washed coal from the tables is distributed to three 54x70 in. Bird centrifuges. The dewatered centrifuge product is combined with the product of the dewatering screens prior to loading into railroad cars.

During the early part of 1951 complaints were received from officials at Republic's coke ovens concerning the moisture content of Russellton washed coal. The records revealed that during the first half of 1951 the shipped washed coal moisture averaged

7.03 pct and that daily moistures fluctuated from 5.35 to 9.67 pct.

The coke oven officials asked for a washed coal averaging less than 6 pct moisture and a top limit of 6 pct moisture. These requirements could not be met, in toto, without reducing the output of washed coal, or, as an alternative, adding additional dewatering and possibly thermal drying equipment to the plant. A compromise goal was finally established of providing a washed coal averaging 6 pct or less moisture, with every effort being made to keep the moisture content of any one day's production below 6 pct. This goal was based on statistics of equipment available in the preparation plant, on some proposed minor alterations in the methods used for dewatering, and on the premise that plant output must be maintained.

The Methods Proposed for Reducing Moisture

The preparation plant dewatering set-up was studied and two methods for improving the results were given consideration. The first method involved spraying oil on the raw $\frac{1}{4} \times 0$ -in. coal prior to water-wetting it ahead of the Deister tables. It was postulated that the oil spray would preferentially coat the coal particles and prevent them from being wetted by water. If the coal surfaces were not wetted, a centrifuge product having a lowered moisture content would be produced. This method was suggested by the experiences of the Lehigh Navigation Co. and the Semet-Solvay Co. The Lehigh Navigation Co. used a Bird centrifuge to dewater froth flotation products in an experimental plant. It was determined that the moisture content of the centrifuge cake varied considerably depending on the reagent used. The Semet-Solvay Co. experimented with Benzol and kerosene on a laboratory scale and with kerosene on a plant scale and determined that reagents of this type would lower the moisture content of a centrifuge cake.

The second method, based on Semet-Solvay experience, involved revamping of the interior design of the Bird centrifuges to allow introduction of relatively clear classifying water which would remove slimes not removed by the classification action that normally takes place in the large diameter end of the centrifuge bowl. Semet-Solvay experience indicated that the centrifuge cake moisture could be reduced by several per cent. This method, obviously, would provide better results in a plant using a positive pulp bleed than it would in one operated with an interior closed circuit. At Russellton, prior to the experimental program, approximately 600 gpm of a slimy pulp was removed from the circulating water system and thickened by means of a 75-ft diam Dorr thickener. The thickener underflow, at the rate of 200 gpm, was pumped to a waste pond at intervals aggregating approximately 20 pct of the total operating time. This underflow contained between 20 and 30 pct solids by weight and served to remove slime solids, thus preventing a buildup of slimes in the system. The thickener overflow ranged between 0.5 and 5.0 pct solids by weight.

The Base Comparison Period

For comparison purposes, the period between Aug. 1 and Sept. 26, 1951, was taken as a standard. The results of the experimental work would be evaluated in terms of similarities to or deviations from the base period. During this period the $3\frac{1}{2} \times 0$ -in.

washed coal averaged 7.17 pct moisture, while the daily moistures fluctuated from 5.34 to 9.02 pct. The $\frac{1}{4} \times 0$ -in. washed coal averaged 13.97 pct moisture, while the daily moistures fluctuated from 11.45 to 15.76 pct. The daily moisture values are tabulated in Table I and presented in graphic form in Fig. 1.

Results of Spraying Oil on Raw $\frac{1}{4} \times 0$ in. Coal

A light oil, Esso No. 45 Coalspray, was being used at Russellton at the rate of 0.1 gallon per ton of washed coal to improve the bulk density of the coke oven charge and to freezeproof the washed coal in transit. The existing spray line was extended into the plant, and the same oil, at the same flow rate, was sprayed on the raw $\frac{1}{4} \times 0$ -in. coal as it discharged from the raw coal sizing screens. The existing flow rate represented approximately 0.3 gal per ton of raw $\frac{1}{4} \times 0$ -in. coal. After being oiled, the coal went through two transfer points prior to entering the water-wetting box providing a mixing of oil and coal.

Preliminary results with the oil spray were encouraging, as can be seen by looking at Fig. 2. The moisture content of the washed coal was reduced and, in addition, the ash content of the table-washed coal went down while the ash content of the table refuse went up. A thin froth consisting of small bubbles was observed on the coal flow portion and discharge side of each table, and it seemed that a solution had been found.

The moisture reduction obtained did not meet the objectives, however, so the amount of oil added was doubled. The first result was an increase in washed coal moisture. Large and numerous bubbles flowed across the table. Bubbles began to form on the surface of the Chance cone and a heavy froth started to build up on the water surface of the Dorr thickener. When the froth on the thickener became several feet thick it was necessary to remove it by means of a power shovel. Then the froth began to plug pipelines in the plant and a soft spot developed in the rubber covering of one of the tables. Since all the conveyors in the plant were either flight or bucket-type, the oil did not harm them but instead served as a lubricant. The plugging of the pipelines, the building up of the heavy froth on the thickener, and the small decrease in moisture, combined, led to abandonment of the oil addition method.

Results Obtained by Using Classifying Water In the Bird Centrifuge

The Bird Machine Co. and the Semet-Solvay Co. had already obtained a moisture reduction of several per cent, as mentioned previously, in the Bird centrifuge dewatered fine coal produced at both the Harewood and Trailee plants of the Semet-Solvay Co. Republic Steel was given the benefit of this experience. Figs. 3 and 4 show the changes made in a standard Bird coal centrifuge to provide the desired classifying action. Fig. 3 shows the 2-in. classifying waterline carried inside the slurry feedline, the new discharge port in the slurry feedline, some of the new feed ports discharging to the spiral, a baffle placed across the cone to direct the classifying water, and the placement of one of the six classifying water nozzles. Fig. 4 shows how the original filtrate ports were altered to lower the pool depth and obtain a better classifying or desliming action.

William Noone, Preparation Engineer with Semet-Solvay Co., reported that it is essential that the classifying water be applied at the point where the

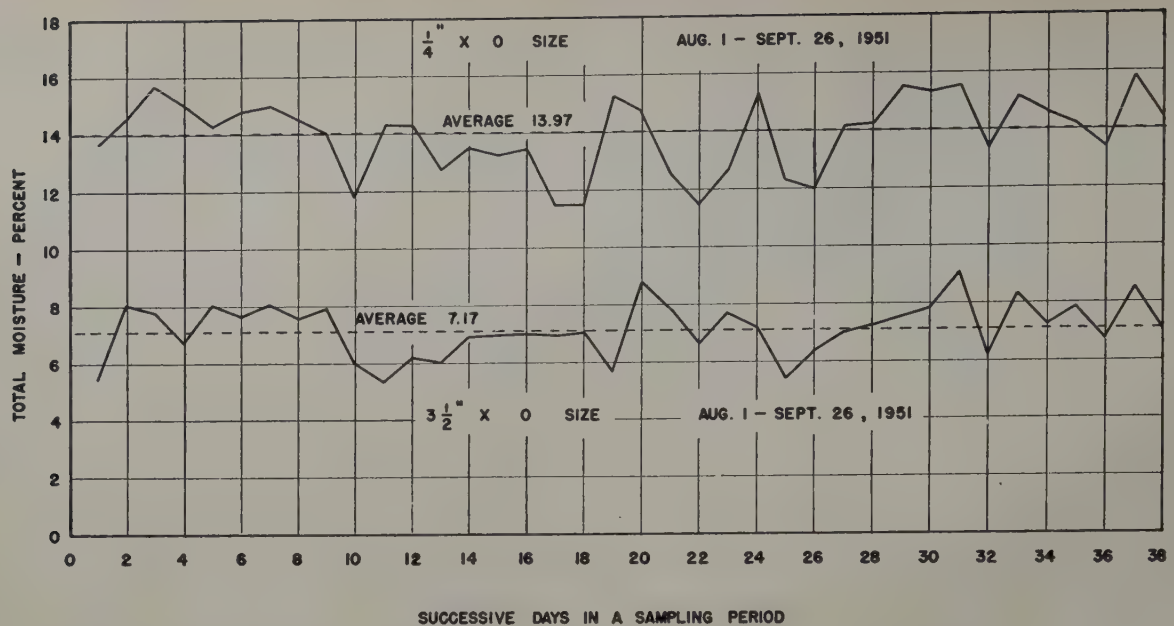


Fig. 1—A graphic presentation of daily moisture values shown in Table I. A base period from Aug. 1 to Sept. 26, 1951, was used as standard.

coal is changing direction as it moves forward under the impetus of the spiral. The change in slope in the centrifuge bowl provides a turning movement of each particle within the mass of the coal particles; therefore the classifying water should be added at or just prior to the change of slope point. Classifying water hitting a mass of coal moving as a constrained unit would have little desliming effect. The turning and rubbing of coal particle on coal particle helps to dislodge slime particles and definitely exposes more surface to water contact.

The normal method of operation at Russellton was and is to operate three Bird centrifuges on a continuous basis. A spare spiral is kept on hand which is used to replace the spiral in No. 1 centrifuge when

its edges are worn down. No. 1 spiral is rebuilt and used to replace No. 2 spiral. Then No. 2 is rebuilt and used to replace No. 3 and so on. With this system of operation it was necessary to convert the centrifuges to the classifying water method on a one by one basis.

Fig. 5 presents in graphic form the moisture data contained in Table I for the period between Nov. 19 and Dec. 26, 1951. During this period No. 1 centrifuge was operated with classifying water, and No. 2 and 3 without it. The water addition represented approximately 70 gpm at 27-lb pressure. There was an immediate decrease in the moisture contents of both the 3½x0-in. and the ¼x0-in. washed coal products. The 3½x0-in. washed coal

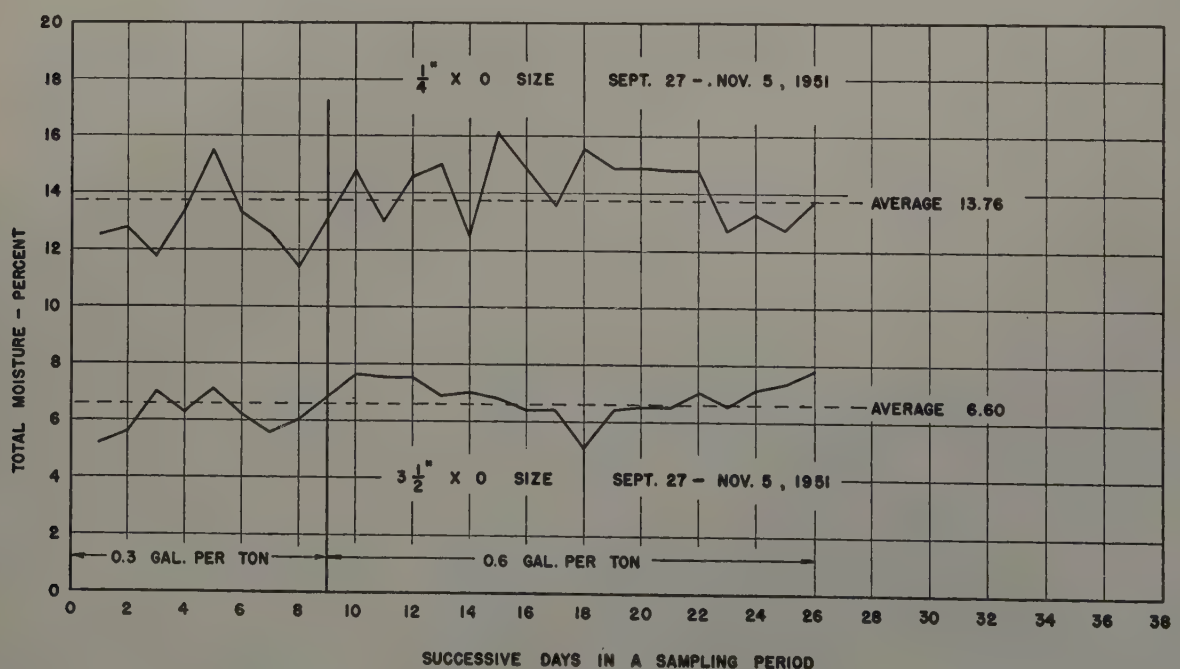


Fig. 2—Oil additive periods. Preliminary results were encouraging.

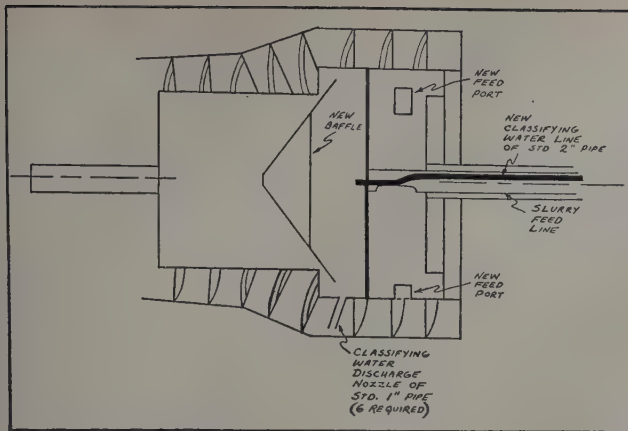


Fig. 3—Changes made in Bird centrifuge to allow introduction of classifying water for removing extra slimes.

averaged 6.54 pct moisture, with fluctuations between 4.49 and 7.84 pct, and the $\frac{1}{4}$ x0-in. washed coal averaged 12.75 pct moisture, with fluctuations between 11.23 and 13.93 pct.

Fig. 6 presents in graphic form the moisture data contained in Table I for the period between Dec. 26, 1951, and Feb. 22, 1952. During this period No. 1 and 2 centrifuges were operated with classifying water additions and No. 3 was operated without classifying water. The water addition represented approximately 70 gpm per centrifuge. During this period the moisture contents of both the $3\frac{1}{2}$ x0-in. and the $\frac{1}{4}$ x0-in. washed coal products decreased. The $3\frac{1}{2}$ x0-in. washed coal averaged 5.78 pct moisture, with fluctuations between 4.74 and 7.28 pct, and the $\frac{1}{4}$ x0-in. washed coal averaged 11.48 pct moisture, with fluctuations between 10.49 and 12.88 pct.

Fig. 7 presents in graphic form the moisture data contained in Table I for the period between Feb. 23 and March 31, 1952. During this period all three centrifuges were operated with classifying water, supplied at the rate of approximately 70 gpm per centrifuge. During this period the $3\frac{1}{2}$ x0-in. washed coal showed a slight decrease in moisture while the $\frac{1}{4}$ x0-in. washed coal contained the same average moisture as in the previous period. The $3\frac{1}{2}$ x0-in. washed coal averaged 5.60 pct moisture, with fluctuations between 4.49 and 6.90 pct, and the $\frac{1}{4}$ x0-in. washed coal averaged 11.48 pct moisture, with fluctuations between 10.31 and 12.66 pct.



Fig. 4—Changes made in Bird centrifuge filtrate ports to allow use of classifying water.

tuations between 4.49 and 6.90 pct, and the $\frac{1}{4}$ x0-in. washed coal averaged 11.48 pct moisture, with fluctuations between 10.31 and 12.66 pct.

Fig. 8 presents in graphic form a summary of the moisture reduction results obtained at Russellton. The $3\frac{1}{2}$ x0-in. washed coal moisture went from an average of 7.17 pct at the start of the testing period down to an average of 5.60 pct at the end of the period. The $\frac{1}{4}$ x0-in. coal moisture went from an average 13.97 pct at the start of the testing period to an average 11.48 pct at the end of the period.

The Effect of Size-Consist on Moisture Content

Fig. 8 also presents in graphic form a summary of the percentages of $\frac{1}{4}$ x0-in. washed coal in the combined $3\frac{1}{2}$ x0-in. washed coal during the major portion of the testing program. The amount of $\frac{1}{4}$ in. showed a marked decrease between the second and third testing periods, i.e., the oil additive and one centrifuge equipped with classifying water periods, and a marked increase between the third and

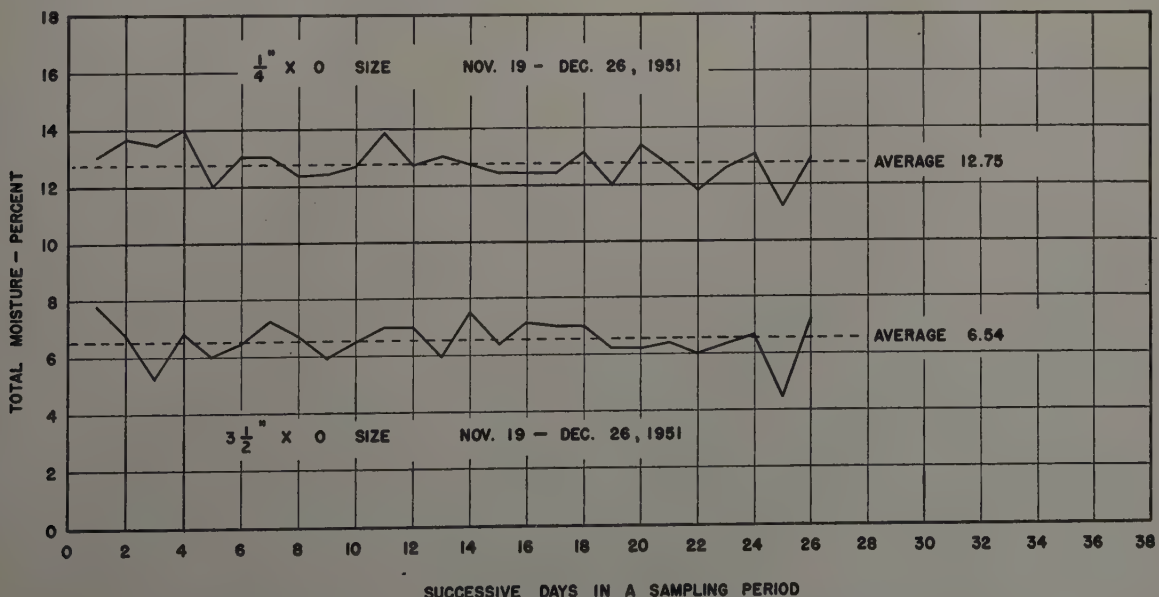


Fig. 5—Period during which classifying water was added to No. 1 centrifuge only.

Table I. A Tabulation of Moisture and Size Consist Values

Sample No.	Base Period		Oil Add Period				No. 1 Centrifuge Back Washed			No. 1 and 2 Centrifuges Back Washed			No. 1, 2 and 3 Centrifuges Back Washed		
	$\frac{1}{4} \times O$ -In., H ₂ O, Pct	$\frac{3}{2} \times O$ -In., H ₂ O, Pct	$\frac{1}{4} \times O$ -In., W.C., Wt., Pct	$\frac{1}{4} \times O$ -In., H ₂ O, Pct	$\frac{3}{2} \times O$ -In., H ₂ O, Pct	$\frac{1}{4} \times O$ -In., W.C., Wt., Pct	$\frac{1}{4} \times O$ -In., H ₂ O, Pct	$\frac{3}{2} \times O$ -In., H ₂ O, Pct	$\frac{1}{4} \times O$ -In., W.C., Wt., Pct	$\frac{1}{4} \times O$ -In., H ₂ O, Pct	$\frac{3}{2} \times O$ -In., H ₂ O, Pct	$\frac{1}{4} \times O$ -In., W.C., Wt., Pct	$\frac{1}{4} \times O$ -In., H ₂ O, Pct	$\frac{3}{2} \times O$ -In., W.C., Wt., Pct	$\frac{1}{4} \times O$ -In., W.C., Wt., Pct
1	13.62	5.46		12.51	5.23		13.07	7.84	30.95	12.60			10.98	5.35	26.82
2	14.51	7.96		12.73	5.62		13.67	6.79	34.15	10.63			10.97	6.90	
3	15.60	7.75		11.81	6.90		13.52	5.18	26.48	12.25	6.75		11.83	5.00	30.02
4	15.11	6.75		13.41	6.34		13.93	6.77	29.63	11.61	6.09		10.90	6.03	31.87
5	14.28	8.02		15.44	7.03		12.03	6.07	32.13	12.78	6.30		11.10	5.13	34.90
6	14.72	7.62		13.31	6.12		13.02	6.44	33.05	12.88	6.78		11.42	4.90	28.35
7	14.98	8.04		12.62	5.63		13.02	7.20	33.05	11.51	6.25		12.28	5.17	30.17
8	14.52	7.56		11.35	6.01		12.32	6.74	30.00	10.50	6.53		11.72	5.02	29.43
9	14.08	7.87		12.98	6.70	32.75	12.42	5.97	28.90	11.12	5.77		11.49	5.44	31.68
10	11.77	6.00		14.73	7.62	33.63	12.61	6.51	30.56	11.25	7.28		11.25	5.13	30.80
11	14.33	5.34		13.01	7.54		13.80	6.95	30.70	10.59	4.74		11.17	4.49	34.50
12	14.17	6.13		14.55	7.52	31.94	12.76	6.97	23.20	10.49	5.65		11.73	6.64	29.15
13	12.74	5.99		14.96	6.88	32.80	12.99	6.00		11.79	5.30		11.08	6.37	34.98
14	13.51	6.83		12.52	7.06	34.35	12.68	7.39	32.00	11.64	6.04	37.25	11.10	5.91	35.40
15	13.23	6.90		16.07	6.77	32.83	12.43	6.46	32.10	11.21	6.24	32.33	11.55	5.78	33.50
16	13.46	6.95		14.83	6.39	30.83	12.43	7.13	28.10	11.33	5.82	27.28	11.49	4.94	36.50
17	11.57	6.89		13.65	6.37	30.43	12.41	6.93	26.35	12.22	6.23	35.55	11.18	5.57	34.60
18	11.45	6.99		15.48	5.14	29.35	13.09	7.05	27.63	11.08	6.04	27.03	12.50	6.56	27.50
19	15.25	5.70		14.93	6.34	32.65	12.01	6.20	21.90	12.18	5.17	27.58	12.66	6.26	29.45
20	14.74	8.70		14.93	6.48	32.23	13.37	6.18	28.33	12.17	6.67	26.93	11.85	6.15	29.20
21	12.52	7.83		14.77	6.53	34.78	12.66	6.36		11.67	6.92	32.28	11.78	5.72	29.78
22		8.07		14.82	7.04	31.95	11.74	6.09	31.65	11.76	5.20	31.78	12.31	6.02	29.97
23	11.59	6.63		13.25	7.14	30.25	12.51	6.32	22.80	11.97	5.64	32.70	11.20	4.73	29.33
24	13.67	7.62		12.69	6.62	34.70	12.98	6.65	26.75	11.45	5.73	31.63	11.20	5.55	31.50
25	15.34	7.15		12.69	7.30	33.20	11.23	4.49	29.50	11.17	6.35	30.92	10.31	5.28	29.58
26	12.26	5.37						7.22	23.90	11.67	6.24		11.35	5.47	28.95
27	12.09	6.34								11.15	6.08	30.28			
28	14.07	6.90								11.82	5.56	31.73			
29	14.18	7.23								11.29	5.36	30.53			
30	15.47	7.53								11.78	5.51	28.88			
31	15.43	7.80								11.99	6.00	31.57			
32	15.58	9.02								10.78	5.69	27.45			
33	13.62	6.23								11.25	5.38	32.38			
34	15.16	8.25								11.16	5.20	33.60			
35	14.68	7.18								10.92	4.98	32.78			
36	14.19	7.86								10.64	4.71	28.22			
37	13.43	6.77								10.66	6.07	31.18			
38	15.76	8.51								11.70		31.95			
39	14.29	6.94								11.14	5.01	29.73			
40											5.83	30.10			
41										11.17	5.97	26.02			
42										11.90	5.71	32.05			
Average	13.97	7.17		13.76	6.60	32.42	12.75	6.54	28.91	11.48	5.78	31.11	11.48	5.60	31.12

fourth testing periods. Periods 2, 4, and 5 showed little variation, however, and it seems safe to assume that the size-consist of the washed coal had little to do with moisture reduction obtained.

The addition of approximately 210 gpm of classifying water to the centrifuges did not add materially to the amount of circulating water in the system. In effect, about 30 gpm of additional fresh water was

required. Some clear makeup water was diverted from other addition points and the remainder of the classifying water was made up of clarified water overflowing the 75-ft diam Dorr thickener. Periodic samples of the thickener overflow showed an increase in its solid content, and to keep the overflow solids content in the range of 5 to 10 by weight it was necessary to operate the thickener underflow

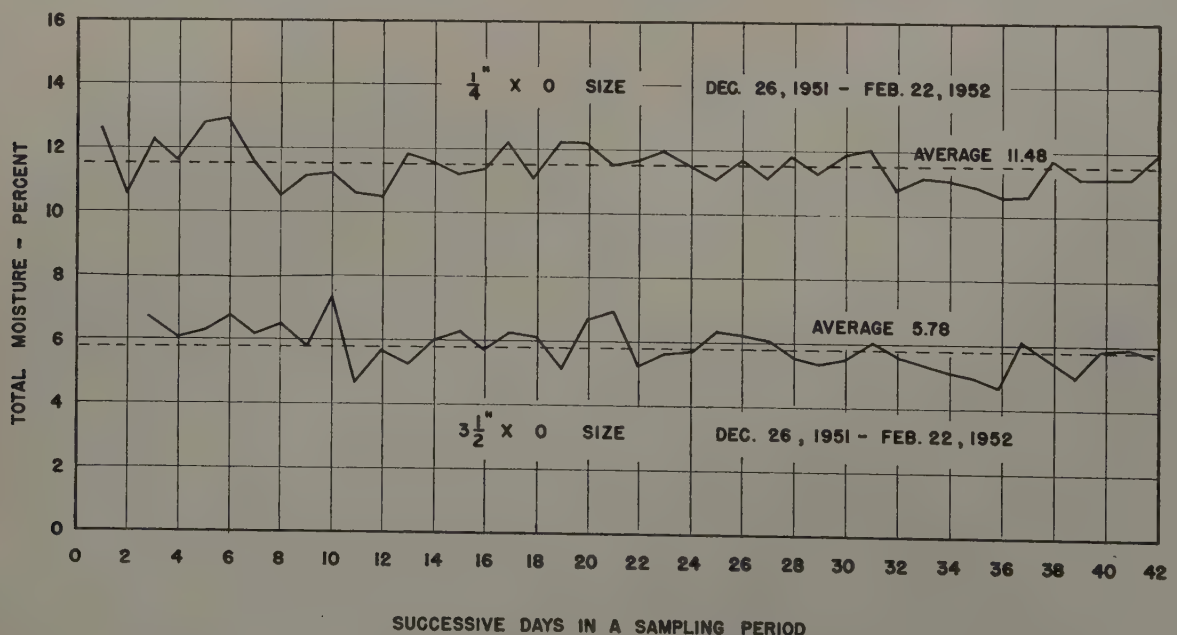


Fig. 6—Period during which classifying water was added to No. 1 and No. 2 centrifuges.

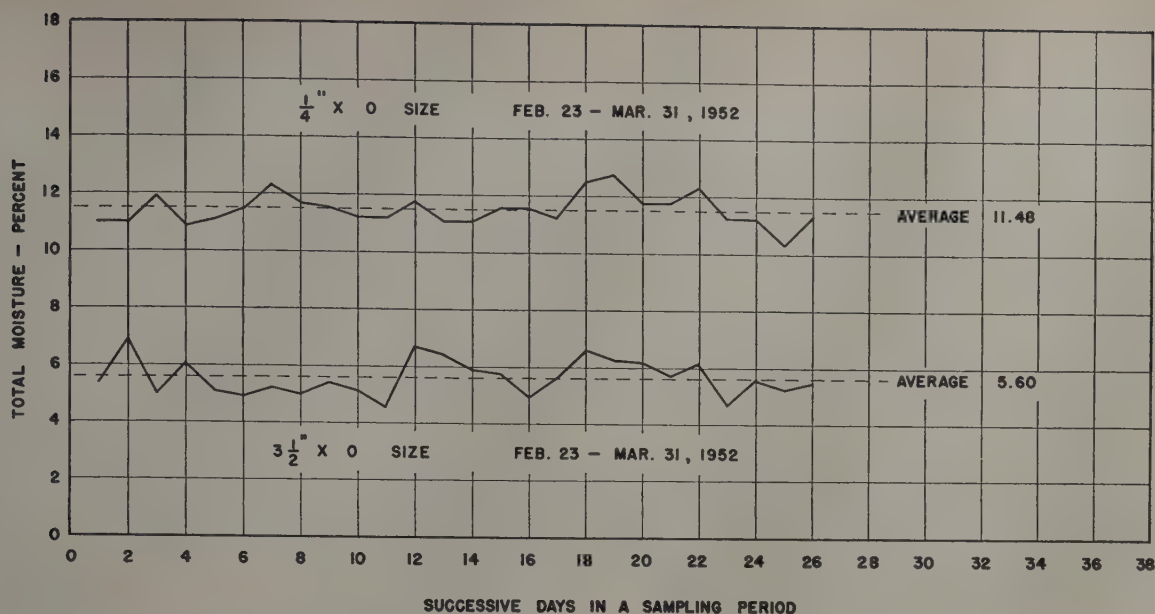


Fig. 7—Period during which classifying water was added to all three centrifuges.

pump approximately 35 pct of operating time, in place of the 20 pct previously reported.

Effect of Classifying Water on Centrifuge Wear

The use of classifying water in the centrifuges increased the rate at which the edges of the spirals wore and also increased wear on other portions of the machine. Prior to use of the classifying water a centrifuge would handle approximately 60,000 tons of $\frac{1}{4} \times 0$ -in. coal before it would have to be overhauled. After the use of classifying water was started a centrifuge would handle only 30,000 tons of $\frac{1}{4} \times 0$ -in. coal before it had to be overhauled. The use of $\frac{1}{2}$ -in. high retaining strips on the inside of the drum allowed a build-up of coal particles,

saved wear on the drum, and increased the wear-life between overhauls to approximately 45,000 tons.

The Semet-Solvay Co. encountered more or less the same wear difficulties and working with representatives of the Bird Machine Co. found that changing the centrifuge speed from 640 rpm to approximately 540 rpm prolonged spiral life and still allowed production of a low moisture content centrifuge cake. It is the intention at Russellton to try the reduced centrifuge speed as soon as possible.

The ash content of the centrifuge cake produced during the base period of Aug. 1 through Sept. 26, 1951, averaged 8.60 pct. The variation in ash content was from 8.00 pct to 9.51 pct. The ash content of the centrifuge cake produced during the final classifying period of Feb. 23 through March 31, 1952, averaged 8.06 pct. The variation in ash content was 7.45 pct to 8.90 pct. It seems obvious that the material being removed by the classifying water was a high-ash slime.

Conclusions

The addition of small amounts of a light oil to $\frac{1}{4} \times 0$ -in. raw Thick-Freeport coal, prior to treatment by means of wet tabling, will not materially reduce the moisture content of the cake produced by a Bird centrifuge. Oil additions were beneficial from an ash reduction standpoint but created so many operating difficulties that the additions had to be abandoned.

The use of classifying water to remove slimes not removed by normal centrifuge classification provided a marked reduction in the moisture content of the cake produced by a Bird centrifuge. The $\frac{1}{4} \times 0$ -in. washed coal moisture averaged 13.97 pct prior to the use of classifying water and 11.48 pct after the use of classifying water. This is a reduction of 2.49 pct. The 2.49 pct reduction in fine coal moisture helped to reduce the moisture content of the composite $3 \frac{1}{2} \times 0$ -in. washed coal by 1.57 pct.

The use of classifying water also reduced the ash content of the cake produced by a Bird centrifuge. The objectives of the moisture reduction program were attained. The one disadvantage was that centrifuge maintenance requirements were increased.

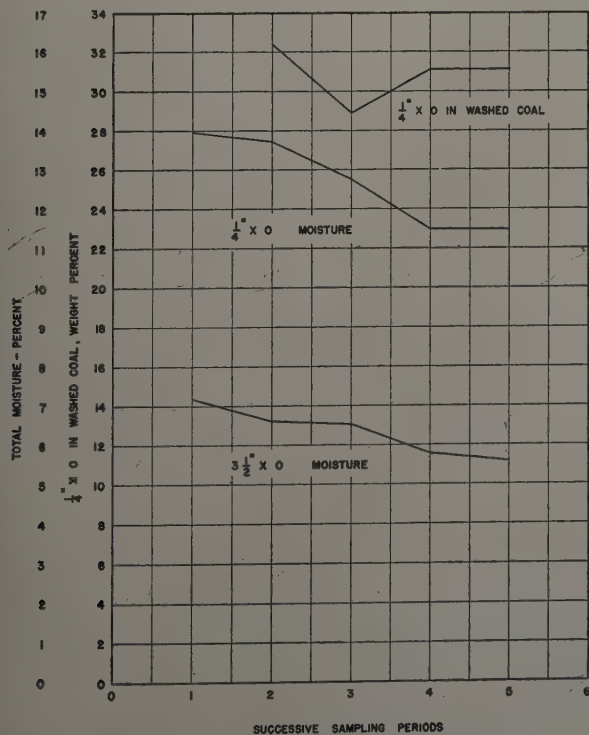


Fig. 8—Summary of moisture results at Russellton.

Providing Large Industrial Water Supplies by Induced Filtration

Water supplies dependent upon induced infiltration can be developed by vertical wells, by infiltration galleries, or by horizontal water collectors. Each method is described and the advantages of water obtained by induced infiltration with respect to cost, chemical quality, and temperature are shown by specific water supplies.

by Fred H. Klaer, Jr.

THE subject of ground water in industry is assuming increasing importance as new processes and new industries are developed. Industrial specifications of chemical quality and temperature for process water and for cooling purposes are becoming more rigid, and water must now have many properties to meet these requirements. As ground water is usually more constant in quality and temperature, many industries are turning to ground-water supplies to satisfy their growing need.

As many published articles have stated during the past several years, the ever increasing demands have caused shortages of water supply in many localities throughout the United States. Although such shortages are generally of a local nature, they have given rise to many attendant problems, such as increased pumping costs, salt water encroachment, and rising ground-water temperatures.

One partial solution to the problems of overdevelopment of the ground-water reservoirs in many areas can be accomplished by developing ground-water supplies that are dependent upon induced infiltration and by planning the use of underground reservoirs. These reservoirs can be filled during periods of excess precipitation and stream flow and can be emptied during periods of drought, in a manner similar to the use of surface reservoirs. In this way water that otherwise would be wasted into the oceans is salvaged during periods of excess.

The development of water supplies dependent on induced infiltration is a relatively new concept and should be defined. When a well is pumped, the water level around the well is lowered in the shape of an

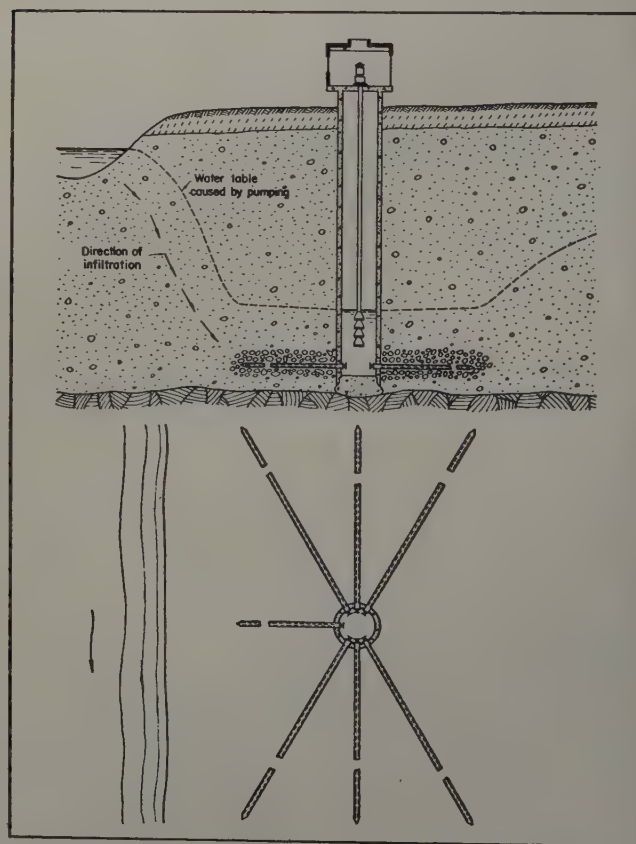


Fig. 1—Horizontal collector designed to induce infiltration.

inverted cone or *cone of depression*. As pumping is continued, the cone of depression will continue to expand, horizontally and vertically, until the quantity of water crossing the perimeter of the cone plus the recharge within the area of the cone is equal to the quantity of water being pumped.

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Discussion on this paper, TP 3525H, may be sent (2 copies) to AIME before Aug. 31, 1953. Manuscript, Oct. 30, 1952. Chicago Meeting, September 1952.

Along most of the larger rivers of the United States and particularly in the northeastern quarter of the country that has been subjected to continental glaciation, alluvial deposits of sand and gravel have been left during glacial and post-glacial times. The materials are gravels, sands, silts, and clays in mixtures of various proportions. These deposits are generally sorted and stratified.

Under natural conditions water falling as precipitation on the earth's surface saturates the surficial materials and moves in a general horizontal direction to drain into some surface stream. If, however, a well, infiltration gallery, or water collector located in the alluvial deposits of sand and gravel adjacent to the stream is pumped, the cone of depression developed by pumping the well will change the hydraulic gradient toward the stream and water will move from the stream toward the well and into the aquifer. This process is called *induced recharge* or *induced infiltration*.

The process of induced infiltration assures the permanency of the water supply, as it derives water from a perennial source and is not dependent on recharge caused by the seasonal distribution of rainfall and the growing season. Even in cases where the surface source of water may become deficient during the dry summer months, water supplies dependent on induced infiltration may be maintained if sufficient ground-water storage is available to produce the required quantities of water during the period of deficient stream flow. Ground-water levels may be lowered and the available ground-water storage deliberately decreased during dry periods, as long as sufficient excess flow is available during the remainder of the year to replenish the storage. The utilization of ground-water aquifers as underground reservoirs, controlled in a manner similar to the control of surface reservoirs, will become an industrial necessity in certain areas as the demands for ground water surpass the natural rates of recharge.

The principles and theory of induced infiltration have been discussed in detail by Meinzer,¹ Kazmann,^{2,3} Rorabaugh,^{4,5} and others, and only a brief discussion is required here. The requisite local conditions for inducing infiltration include a surface source of water such as a stream or lake that is connected hydraulically with an aquifer. The aquifer may be a consolidated formation but is usually a deposit of permeable sand and gravel open to the stream so that water may flow in either direction between the surface source and the aquifer.

When a well is pumped, the shape of the developed cone of depression in the vicinity is controlled largely by the permeability of the formation, the quantity of water being pumped, and the total available head. Under homogeneous conditions the cone of depression is circular in shape and symmetrical around the well. Ground water flows toward the well equally from all directions if the permeabilities of the aquifer, the hydraulic gradients, and the areas through which the water passes are uniform.

If, however, a well is placed near a perennial stream that is interconnected hydraulically with the aquifer, the cone of depression will become distorted with the steeper slopes toward the stream, and water will flow from the stream into the aquifer to replace that which has been pumped out. Since sufficient recharge is available from the perennial stream, the cone of depression toward the stream will eventually assume a relatively stable shape. The ultimate

quantity of water that can be obtained can be determined in advance by a detailed hydrogeological survey.

Methods of Developing Water Supplies By Induced Infiltration

The principal methods of developing water supplies by induced infiltration are those normally used in the collection of ground water, namely, vertical wells, infiltration galleries, and horizontal water collectors. For each method there are advantages and disadvantages as to initial cost, efficiency yield, operation, and longevity.

Vertical wells have been used successfully for many years, especially along the larger streams in the United States such as the Ohio River in Pennsylvania⁶ and West Virginia.^{7,8} The relative cost of a vertical well system to induce infiltration is often lower than other methods, but such a system has certain inherent disadvantages from the standpoint of efficiency, maintenance, operation, and longevity.

The cone of depression developed by all wells is made up of a head loss due to the movement of water through the aquifer to the well plus an additional loss in head required to move the water through the screen and to the pump intake. This latter so-called *well loss* factor is believed to be due in part to the change from laminar to turbulent flow in the immediate vicinity of the well. In some wells, the well loss factor may constitute more than half the total drawdown. In formations of limited thickness such as the alluvial and glacial terrace deposits along many of the streams of the United States, a system of vertical wells can develop only a fraction

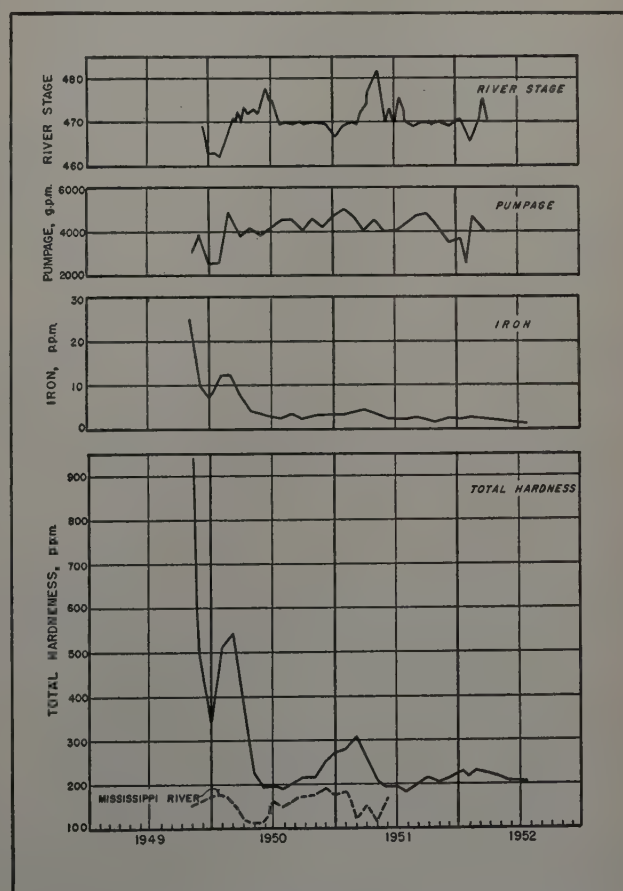


Fig. 2—River stage, pumpage, iron content, and total hardness of water from horizontal collector at Central Fiber Products Co., Quincy, Ill.

of the total head available in the aquifer. Under uniform conditions of permeability and thickness the cone of depression developed by a vertical well is roughly circular. Although some distortion of the cone is caused by induced infiltration, the actual quantity of water that can be developed by a vertical well is limited.

The oldest method of inducing recharge is probably the infiltration gallery, which has been in use since the days of early Greece. The most famous infiltration gallery in this country is probably that used by Des Moines, Iowa.⁹ Infiltration galleries usually intercept more water than wells, especially in shallow aquifers, and cost less to operate and maintain, but they are considerably more expensive to construct.¹⁰ From a practical economic standpoint the depth to which infiltration galleries can be constructed is limited. They may consist of tunnels, walled conduits with many openings, or a pipeline with open joints or openings in the pipe similar to the many types of strainers used in filter or in vertical wells. They are constructed in unconsolidated materials below the water table and seldom exceed 20 ft in depth. They may be constructed either parallel or normal to the source of recharge.

A relatively new device for obtaining large supplies of water by induced infiltration is the horizontal water collector, see Fig. 1. A large reinforced concrete caisson is sunk to the bottom of unconsolidated deposits of sand and gravel adjacent to a stream. The bottom of the caisson is then plugged and through ports near the base of the caisson horizontal screens are projected to distances of several hundred feet into the aquifer, similar to the spokes of a wheel. The screens are projected by a combination washing and jacking process, a large part of the finer material being removed and a natural gravel wall being developed in place around each horizontal screen.

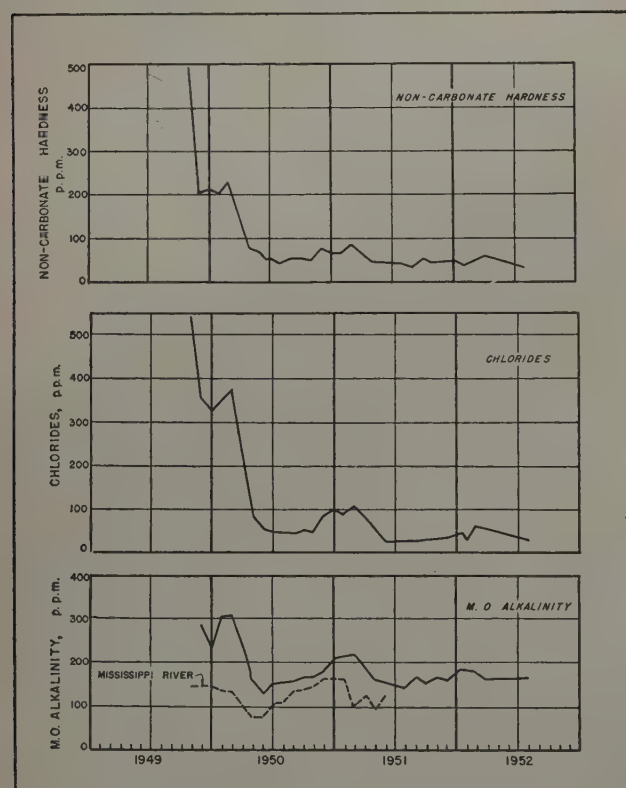


Fig. 3—Non-carbonate hardness, chloride content, and M. O. alkalinity of water from horizontal collector at Central Fiber Products Co., Quincy, Ill.

A horizontal collector is in reality a special type of infiltration gallery in that the pattern of projecting the horizontal screens can be controlled to develop a long section of aquifer parallel to the source of induced infiltration. The horizontal collector can be constructed economically to much greater depths than the average infiltration gallery, thereby utilizing the full available head in the aquifer. Because of the large amount of screen that can be projected, as much as 2000 ft of pipe in some collectors, the entrance velocity of water entering the screens is extremely low, thereby minimizing sudden changes in pressure and well loss and eliminating encrustation. The yields of individual collectors may exceed 20 million gal of water per day under favorable hydrogeological conditions.

There are at the present time more than 100 horizontal collectors in operation or under construction in the United States and about 20 in Europe. The yields of the individual unit range from less than 1 to more than 20 million gal per day. The life of a horizontal collector is as yet unknown, although several units have been in operation for 10 to 15 years without loss of capacity.

Industrial Advantages of Water Supplies Dependent on Induced Infiltration

In the planning of an industrial water supply, several factors must be considered. The ultimate aim in such planning is to obtain a permanent dependable supply of suitable quantity and quality at the lowest unit cost, including not only initial capital cost but also costs of operation and maintenance. The capital cost of a given type of water supply is generally considerably less during the full life of the installation than the costs of operation and maintenance.

Water supplies dependent upon induced infiltration are permanent in that the quantity available to infiltration is usually in excess of the quantities of water required. It is true that the flow of surface streams may vary within wide limits, but it is possible to estimate within close limits the quantity of water that will be available during any given year. The records of stream flow in the United States, collected by the U. S. Geological Survey, the U. S. Corps of Engineers, the U. S. Weather Bureau and many state agencies, generally provide an adequate estimate of the quantity of water that will be available over a period of years. Since infiltration can be induced into underground reservoirs during periods of excess stream flow, these same reservoirs can be drawn on during periods of deficient stream flow to provide from storage the required quantities of water.

The unconsolidated deposits of sand and gravel along many streams may be considered as large natural filters, comparable in many ways to very large slow sand filters. The water obtained by induced infiltration, having been filtered naturally through filters far larger and thicker than any made by man, and at much slower rates of flow, will be free from all turbidity or colloidal and suspended solid matter. The rates of filtration and the velocity of movement through the natural filters are extremely slow and removal of all solids is complete.

Experience, especially in the use of horizontal water collectors, has shown that all organic matter and pathogenic bacteria are removed from water by induced infiltration. Although the removal is not

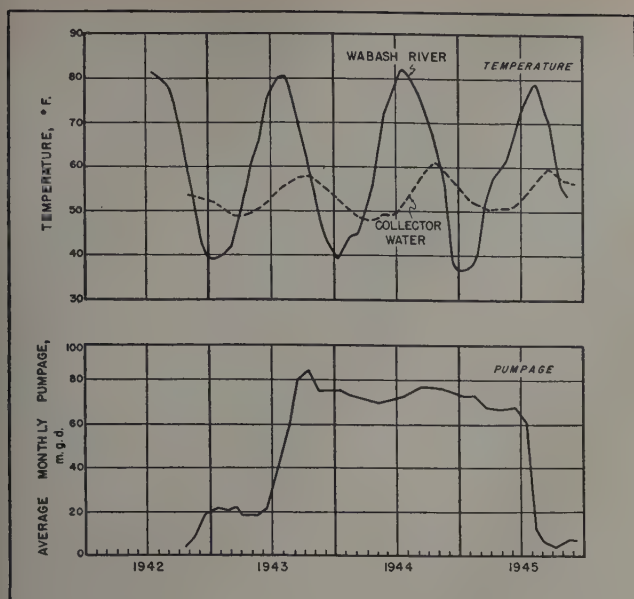


Fig. 4—Temperature of Wabash River and of water and pumpage from horizontal collectors at Wabash River Ordnance Works, Clinton, Ind.

yet fully understood, it is believed that this is accomplished by a thin *filter scum* layer that forms on the stream bottom and that the removal is accomplished in part by oxidation and in part by bacterial action. In recent experiments in California on sewage reclamation by spreading basin infiltration Stone and Garber¹¹ have found that under aerobic conditions of excess oxygen almost complete destruction of organic matter was accomplished after only 7 ft of filtration. These experiments, although not conclusive with regard to induced infiltration from streams, suggest that the major factor in the destruction of organic matter is probably oxidation. This destruction is sufficiently effective to remove even phenols from water pumped from horizontal collector installations along the Ohio River.^{2, 12}

Throughout a large part of the industrial area of the United States the ground water is generally more mineralized than the water in the surface streams, being higher in total hardness, iron, sulphate, chloride, and alkalinity. Water obtained by induced infiltration is a mixture of normal ground water and surface water, and in many localities the chemical quality of induced infiltration water will approach the average chemical quality of the surface stream. Records of chemical quality of water from a horizontal collector of the Central Fibre Products Co. at Quincy, Ill., over a period of more than two years show the changes in chemical quality that may occur, see Fig. 2. The collector, located on the east bank of the Mississippi River, is about 80 ft deep, and the horizontal screens are projected approximately 50 ft below low river level. The collector has a design capacity of 7.5 million gal per day, although the actual pumpage has averaged about 6.8 million gal per day. The original water pumped, which was normal ground water, contained 945 ppm total hardness, 1010 ppm chlorides, 450 ppm alkalinity, and 25 ppm iron. Pumping of the collector was started in November 1949, and the quality of the water improved markedly during the first few months. By June 1950 the iron had dropped to 3 ppm, the total hardness to 200 ppm, the non-carbonate hardness from about 500 to 50 ppm, the chlorides to 50 ppm, and the alkalinity from about 290 to 150

ppm, Figs. 2 and 3. It is apparent from the trend in changes that the ultimate quality of the collector water will approximate the average quality of the river water.

The importance of temperature and humidity control to many industries needs little comment, and the use of water as a cooling medium is becoming increasingly important. Since the temperature of normal ground water is relatively constant and considerably lower than the temperature of air or surface water during the hot summer months, ground water is usually preferred for industrial cooling purposes when available. The temperature of water obtained by induced infiltration will fluctuate in a manner similar to the fluctuation of temperature of the stream source. However, the range in fluctuation will be considerably smaller and the maximum temperature of induced infiltration water will lag the maximum temperature of the surface stream by as much as three months. At the Wabash River Ordnance Works, Clinton, Ind., where a system of six horizontal collectors produced more than 80 million gal of water per day, see Fig. 4, the temperature of the Wabash River fluctuated generally between 37° and 82°F. The temperature of the collector water, however, ranged between 47° and 61°F, even though the collectors were located relatively close to the river. The maximum temperatures of collector water occurred generally about 1½ months after the maximum temperatures of the Wabash River.

Rorabaugh in a recent paper on stream bed percolation in water-supply development⁵ presents data on the operation of a horizontal water collector at the National Carbide Co. at Louisville, Ky. The information he presents on the temperature of the water produced is of especial interest and will be discussed briefly, see Fig. 5. The collector, located on the east edge of the Ohio River, extends about 40 ft below pool stage of the river and about 72 ft above pool stage, so that the pump floor is above flood level. The collector was designed to produce 3.5 million gal a day but has been pumped at a somewhat smaller rate during the past few years. Thermocouples were installed in at least three lateral screens and records were kept of the temperature of the water from the Ohio River, the water from the collector, and the water discharged from individual laterals extending toward the river, upstream parallel to the river, and away from the river. These data are summarized in Table I.

This shows conclusively that the aquifer is an effective heat-exchange unit and that its effective-

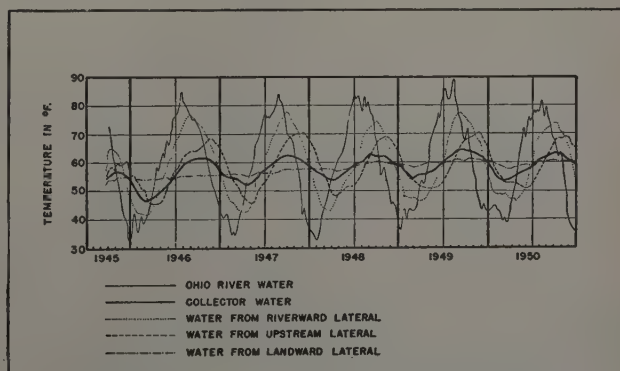


Fig. 5—Temperature of Ohio River and of water from various horizontal lateral screens in horizontal collector at National Carbide Co., Louisville, Ky. Compiled by M. I. Rorabaugh, U. S. Geological Survey.

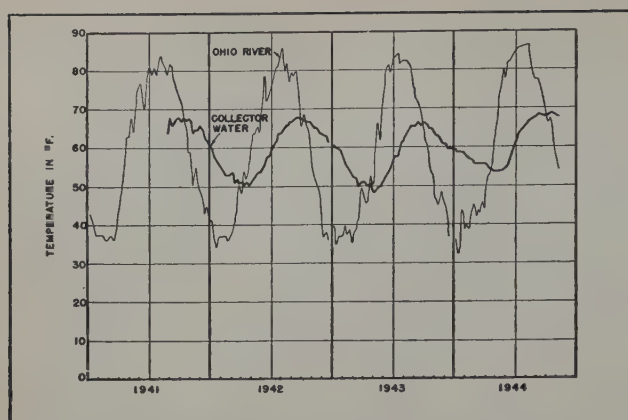


Fig. 6—Temperature of Ohio River and water from horizontal collectors at Indiana Ordnance Works, Charlestown, Ind.

ness can be increased if the distance to the effective line of infiltration is increased. This allows the industrial user to design his water supply to suit his particular requirements.

Similar data for a horizontal collector system at the Indiana Ordnance Works at Charlestown, Ind., have been presented by Kazmann.² This development consisted of seven horizontal water collectors spaced along the north bank of the Ohio River at intervals of 1300 to 2000 ft. The design capacity of the complete system was about 70 million gal of water a day. The system however was pumped at rates of 28 to 56 million gal per day. Although the temperature of the Ohio River ranged between 86°

Table I. Comparison of Water Temperatures

Source of Water	Maximum Temperature, °F	Minimum Temperature, °F	Approximate Lag in Months
Ohio River	89	33	
Riverward lateral	78	42	1
Upstream lateral	70	45	4
Landward lateral	61	55	
Combined discharge of collector	65	47	2-3

and 33°F during the four-year period 1941 to 1944, see Fig. 6, the temperature of the water discharged from the collectors into the storage reservoir ranged from about 69° to 48°F. The maximum temperatures reached by the collector water occurred generally about 2½ months after the maximum temperatures of the Ohio River. Many similar differences have been noted in other installations.

It should be pointed out that infiltration into the ground-water reservoirs cannot be induced in all localities, as in many areas the geologic and hydrologic conditions are not suitable. A detailed hydrogeological survey will determine prior to construc-

tion whether infiltration can be induced and what the capacities of the collecting system will be.

Acknowledgments

The writer wishes to express his appreciation to Henry B. Weiss, Plant Manager, Central Fibre Products Co., Quincy, Ill., and to J. Carl Bode, President, National Carbide Co., New York, for permission to present operational data from horizontal collectors at their Quincy, Ill., and Louisville, Ky., plants, respectively. He also wishes to acknowledge the careful work of M. I. Rorabaugh, U. S. Geological Survey, Louisville, Ky., in compiling data from the National Carbide Co., Louisville, and the work of R. G. Kazmann, Consulting Engineer, Stuttgart, Ark., in compiling data from the Central Fibre Products Co., Quincy, Ill., the Wabash River Ordnance Works, Clinton, Ind., and the Indiana Ordnance Works, Charlestown, Ind.

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Correction

In the December 1951 issue: TP 3194B. Basic Laboratory Studies in the Unit Operation of Crushing, by J. W. Axelson, J. T. Adams, Jr., J. F. Johnson, J. N. S. Kwong, and E. L. Piret. P. 1068, col. 2, par. 2 should read: Rittinger's law is probably too simple a concept to encompass a complicated phenomenon and may very well not be a particularly desirable base or starting point for the development of a full understanding of crushing.

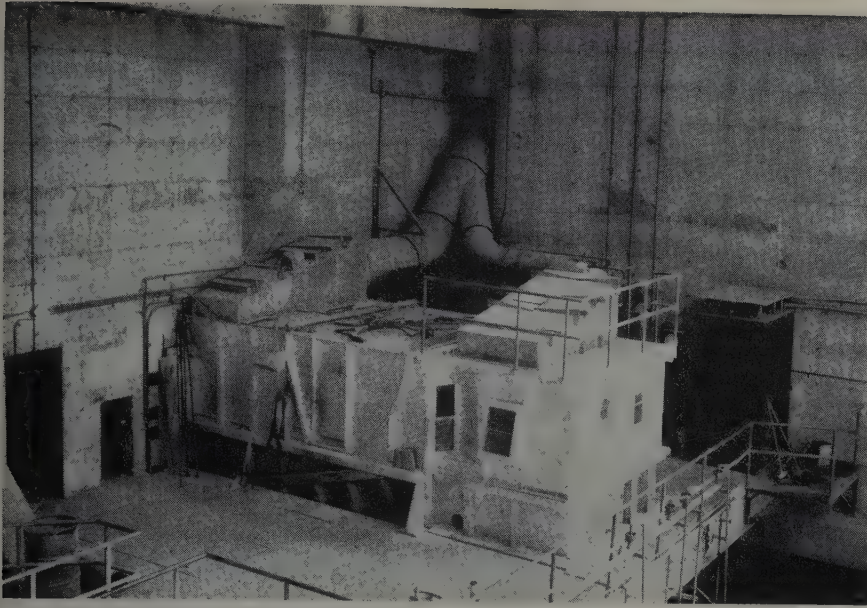


Fig. 1—Pan-feeder and jaw-crusher feed enclosure at the new Ray crushing plant. Photograph shows exhaust hoods and ducting.

Crushing Plant Dust Control at The Ray Mines Division, Kennecott Copper Corp.

by John F. Knudsen

Following the flowsheet from the primary crushing plant of Ray mines division through the final products sample mill, there are eight exhaust systems ranging in size from 12,000 to 30,000 cfm. Where possible, hood face velocities are kept around 500 ft per min, permitting branch velocities of 2500 ft per min. Tight-fitting enclosures are provided over all ore drops, crushing operations, and vibrating screens. All enclosures are exhausted either primarily or secondarily, except those around conveyor head-pulleys where the ejector effect of the falling ore creates sufficient indraft to abate dust dissemination.

IN 1947 an Industrial Hygiene Department was organized to represent and assist the four western mining divisions in industrial hygiene problems. Department headquarters are located at the Utah Copper Division, Magna Plant, Magna, Utah, with a branch office at the Chino Mines Division, Hurley, N. M. The headquarters staff includes a director, a department secretary, an industrial hygienist, a supervising ventilation engineer, and two ventilation designers. To be in closer association with work at the Southwest properties, a field engineer at the branch office conducts field studies which include ventilation problems at the New Mexico and Arizona properties, i.e., the Chino and Ray mines divisions.

Functions of the department are to discover and evaluate existing hazards and to develop protective

measures, either by personal protective devices or by proper ventilation, depending on the nature of the contaminant. Other than dust counting, all air samples collected at the four properties are processed at the home office. If through plant surveys or by suggestion of management some type of ventilation or air conditioning is required, the problem is usually handled by the ventilation section of the headquarters department, where general arrangement or line drawings are prepared, showing hoods, ducting, and equipment. Some hood and special parts detailing is done at headquarters; otherwise, required detailing is done by the respective plant engineering departments.

Prior to the design of industrial ventilation, either new projects or the revamping of old ones, a thorough survey is made of existing conditions. In so far as possible weighted average exposures are determined and adequate ventilation provided on the basis of maximum allowable concentrations of the contaminant in question, as well as medical case histories, if these are available.

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Discussion on this paper, TP 3566B, may be sent (2 copies) to AIME before Sept. 30, 1953. Manuscript, Oct. 14, 1952. Revised April 10, 1953. Los Angeles Meeting, February 1953.

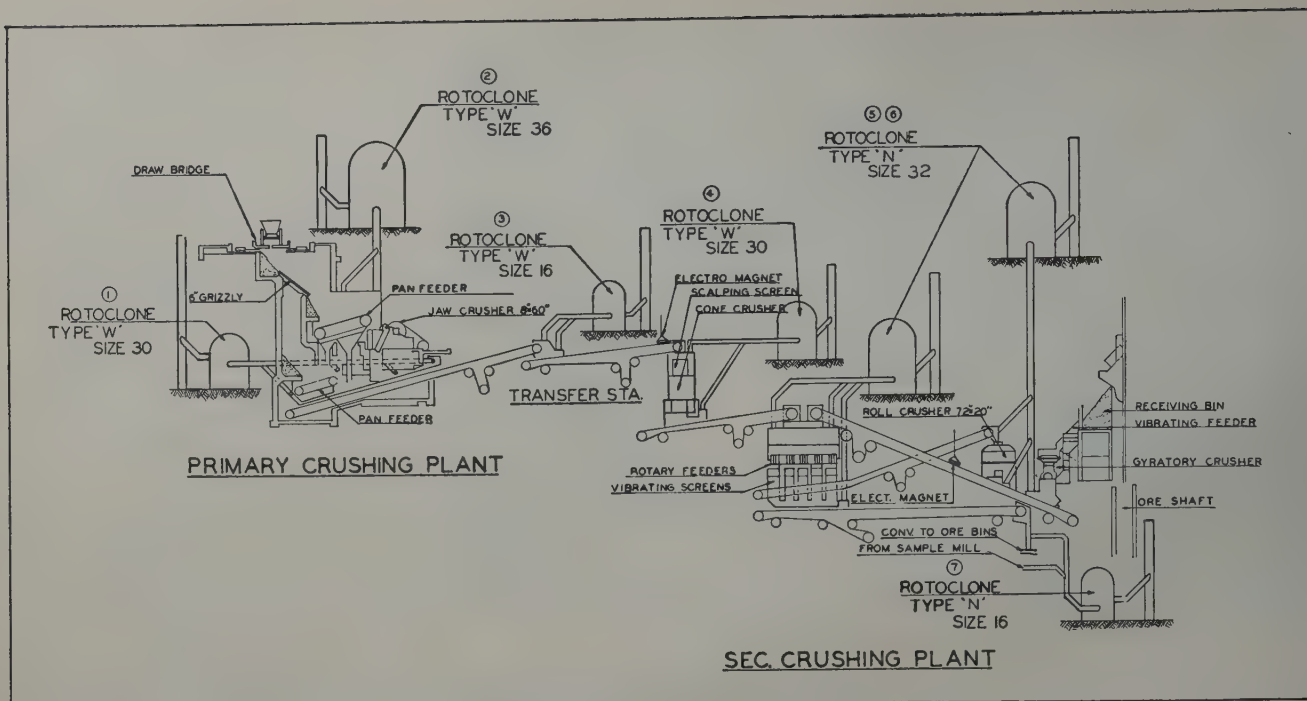


Fig. 2—Diagram of exhaust systems from primary crushing plant through final crushed product sample mill.

Until Kennecott's new Utah refinery came into the picture the major products of Kennecott Copper Corp. Western Mining Properties were copper concentrates and smelted copper; the principal concern of the Industrial Hygiene Department, therefore, was to control dusts in the mines and crushing plants. This was accomplished by dilution or by exhaust ventilation. In the crushing plants at Ray, exhaust ventilation was employed exclusively.

When the decision was made to augment the ore supply from underground with that of ore from open cut operations, it became evident that existing crushing plant facilities were inadequate to process the additional tonnage. As an underground operation the mine produced about 5000 tons of ore per day, whereas under the new plan the tonnage was to be stepped up to 15,000 tons per day, 12,000 tons from open pit operations and 3000 from the underground workings. The crushing is done on two 8-hr shifts at approximately 900 tons per hr.

Preliminary to the design of exhaust ventilation for the Ray property, rafter dust samples collected from the old crushing plant were analyzed and sent to dust collector manufacturers, who determined the settling rate, the particle size, and the collector efficiency ratings.

The main orebody consists principally of schist with intrusions of diabase. Chalcocite is found in the schist, intermingled with pyrite in ribbonlike stringers throughout the workings. Soluble copper, in varying degrees throughout the mine, is mainly in the form of copper sulphate. The percentage of soluble copper salts increased when operations in the open cut mine were begun. This created a problem of corrosion in the dust collectors, much more pronounced than was anticipated from knowledge gained by samples of rafter dust collected from the old crushing plant.

The schist type of ores were found to be water repellent, and when finally wetted became very adhesive, sticking to the conveyor belts and idlers. The dribble problem that ensued was most difficult to

cope with because of the number and length of the conveyors. Obviously this characteristic ruled out the possibility of using any form of water spray preliminary to and during crushing operations.

Exhaust System Design Characteristics

Estimating of air volumes to be exhausted from enclosures and housings continues to be more of an art than a science, although several engineers in the field have made some progress toward standardization. In a paper published in 1949 an attempt was made, using water droplets, to determine the extent of displaced air caused by material falling into a tank.¹ The empirical formulas developed in this paper are not directly applicable for use in the field but were a start in the right direction, to be followed by a more practical plant operations approach.

On large-scale operations, where many variables exist, past experience and field experimentation constitute the principle criteria for establishing required exhaust volumes. Ideas of enclosing the

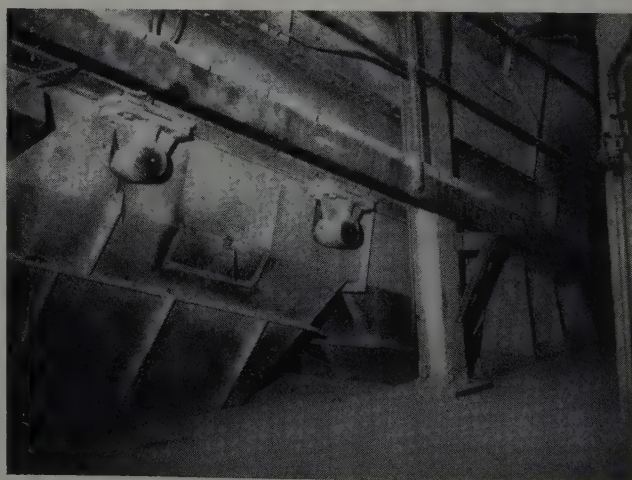


Fig. 3—Grizzly undersize product pan-feeder enclosure, showing access door and recessed lighting.



Fig. 4—Conveyor enclosure between crusher and pan-feeder discharges. Note distance of the exhaust point from the conveyor.

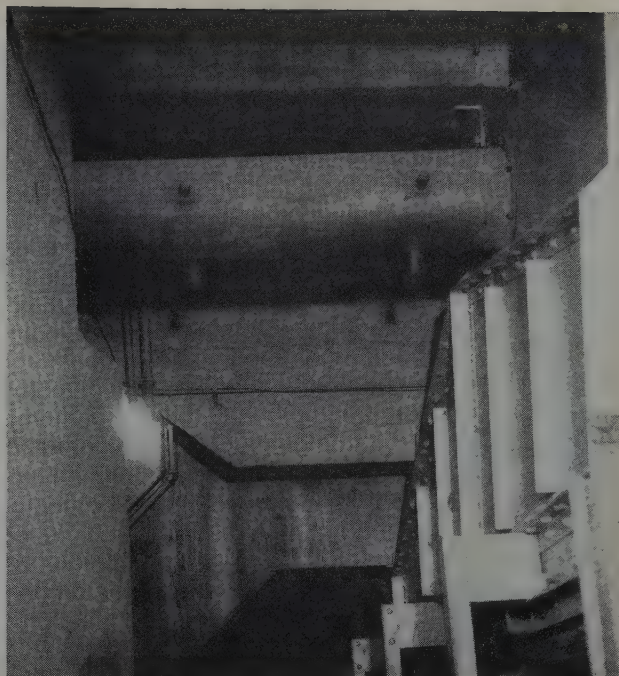


Fig. 5—Final take-off point on conveyor enclosure. The conveyor covers approximately 15 ft beyond the last dumping point.

many different types of machines and locating the exhaust hoods vary extensively, depending on the type of machine, the need for accessibility by the operators and maintenance crews, and the available space around the equipment for housing, hooding, and provision of adequately sized ducting. When exhaust ventilation is engineered conjointly with the design of the plant the pitfalls of inadequate space are overcome, but the situation is complicated by many unknown factors that cannot be tested before completion of the final installation. Some flexibility must be provided to cope with minor discrepancies in design.

Air flow distribution in exhaust systems can be accomplished in one of two ways, by orifices or dampers or by properly adjusted pipe sizes. Although employment of the first method is widespread, there are disadvantages such as rapid wear due to restricted area, tampering by unauthorized personnel, and often the complete removal of dampers. In so far as continued successful operation is concerned, the use of regulated pipe sizes is the better of the two methods. On Ray installations this method was employed on all dust control systems with the exception of the sample mill exhaust system, where diversified control was advantageous. With minor exceptions the system for balancing the air flow in the exhaust systems closely followed the method outlined by Drinker and Hatch.² Ducting loss calculations were based on a compilation of data taken from several sources³⁻⁶ as well as field data developed by the ventilation department.

Wherever possible, hoods were designed so that entering face velocities could be kept under 500 ft per min. Otherwise the openings were located a suitable distance above the dust source to avoid inspiration of large particulate matter. Branch duct velocities could thus be kept in the neighborhood of 2500 ft per min. A good exhaust system is one maintaining sufficient indraft at the enclosure openings

to prevent dust dissemination but transporting only floating dust through the piping system.

Exhaust System Installations

Opinions vary among the authorities in the field of dust control regarding the size of exhaust systems. However, it is the policy of this department to keep the installations small for greater flexibility. The design of the Ray primary and secondary crushing plants was so organized that the various crushing and screening processes were amenable to the smaller exhaust systems. Typical dust control installation is illustrated in Fig. 1.

Following the flowsheet, as shown in Fig. 2, from the primary crushing plant through the final crushed product sample mill, there are eight exhaust systems installed to date, with additional exhaust proposed for the underground ore-crushing operations. The systems range in size from 12,000 to 30,000 cfm.



Fig. 6—Primary screen enclosures, showing the hoods and the ducting.



Fig. 7—Symons crusher area conveyor enclosures, exhaust points about 4 ft off conveyor. With this arrangement, a vertical riser precedes a horizontal run, reducing the possibility of material being projected into the branch ducting.

The ore from the open pit mine is transported to the crusher by 22-cu yd Euclid trucks. A two-gate horizontal slide portal admits the ore to the receiving hopper wherein grizzly bars separate the fines from the oversize product. The oversize material is carried by pan-feeder to the large jaw-crusher with the undersize, minus approximately 7-in. material passing to the undersize bin where it is fed to the No. 1 conveyor by a variable speed pan-feeder. Provisions were made for eventual exhaust from the receiving bin, if found necessary. However, secondary ventilation is provided by exhaust from the enclosure over the oversize pan-feeder and jaw-crusher and by ejector effect of fines passing through the grizzly section.

Table I. Primary Crushing Plant Exhaust Systems

Pan-Feeder, Jaw-Crusher Enclosure, System No. 1		
Hood Location	Hoods, No.	Volume in CFM
Pan-feeder	1	22,500
Jaw-crusher feed	1	7,500
Total	2	30,000
No. 1 Conveyor Loading Point Enclosure, System No. 2		
Hood Location	Hoods, No.	Volume in CFM
Tail of conveyor housing	1	2,500
Undersize feeder discharge	1	5,500
Crusher product feed	1	8,000
Head end conveyor housing	1	4,000
Total	4	20,000

Two ventilation systems, see Table I, are in operation over the 16-hr crushing period, one exhausting the enclosure over the pan-feeder and the jaw-crusher feed opening and one exhausting the feed points enclosure over the No. 1 conveyor. Fig. 1 shows the enclosure over the jaw-crusher feed opening and over the pan-feeder and jaw-crusher with exhaust hoods immediately in front of the portal through the curtain wall separating the receiving bin from that of the crushing plant proper.

As indicated in Figs. 3 and 4, the undersize material pan-feeder, as well as the discharge, is completely enclosed, with exhaust take-off points sev-

eral feet above contact with the conveyor belt. All enclosures are well lighted, lights being housed in recessed receptacles to minimize breakage from flying rock. As illustrated in Fig. 1, operations are visible through windows provided on three sides. Access to the enclosure is provided by two sets of hand-lifted doors and one roll lift section, activated by air, for removing timber and other extraneous materials. Fig. 5 shows the final take-off point on the housing over the conveyor handling the final product leaving the crushing plant. The jaw-crusher discharges a product which is no larger than 10 in., one dimension.

Type W Rotoclones with sludge ejectors are used as a combined exhaust collector; specifications are given in Table IV.

Transfer Station

The transfer station is furnished with exhaust ventilation both for protection of equipment and workmen, although men are not steadily employed at this location. The transfer point is completely housed; a type W Rotoclone, with exhaust hoods at the tail and head of the ore fall, exhausts 4000 cfm from the head and 1000 cfm from the tail of the enclosure.

Secondary Crushing Plant

As shown in the flowsheet, there are four exhaust systems in operation in the secondary crushing plant, with additions planned for the system exhausting the underground ore-receiving bins. When the system is completed, a volume of 99,300 cfm will be exhausted from the crushing and screening operations in the secondary crushing plant.

Symons Crushers Area: Ore from the primary crusher enters a distribution box immediately above the primary or scalping screens. Two -5×8 -ft screens separate the ore into ± 2 -in. products. The fines are conveyed directly to the surge bin and the oversize material passes into a hopper over each of two 7-ft cone crushers. The crushers reduce the ore to about $1\frac{1}{2}$ -in. product.

The head pulley and primary screens are completely housed, see Fig. 6, air being exhausted from



Fig. 8—Secondary screen enclosures, with part of surge bin and feed chutes shown, east side.

each screen housing. No air is exhausted from the enclosure over the head pulley because the ejector effect of the falling ore creates sufficient indraft around the pulley to abate escaping dust. The hopper above each 7-ft cone crusher which receives the screen oversize as well as the area surrounding the cone is housed, but no exhaust is provided. Second-

Table II. Secondary Crushing Plant Exhaust Systems

Symons Crushers Area, Exhaust System No. 4

Hood Location	Hoods, No.	Volume in CFM	
		Each	Total
Primary screen enclosure	2	4,000	8,000
Tail of conveyor enclosures	2	2,000	4,000
Head of conveyor enclosures	2	5,000	10,000
Total	6		22,000

Screening Tower, Exhaust System No. 5

Hood Location	Hoods, No.	Volume in CFM	
		Each	Total
Surge bins	2	2,250	4,500
Tail oversize conveyor enclosure	2	1,500	3,000
Head oversize conveyor enclosure	2	2,000	4,000
Tail undersize conveyor enclosure	1	2,000	2,000
Head undersize conveyor enclosure	1	3,000	3,000
Between undersize screen chutes	8	750	6,000
Total	16		22,500

Rolls and Gyratory, Exhaust System No. 6

Hood Location	Hoods, No.	Volume in CFM	
		Each	Total
Rolls feed hoppers	2	1,400	2,800
Head rolls discharge	2	5,000	10,000
Tail rolls discharge	2	1,500	3,000
Head gyratory discharge	2	3,000	6,000
Tail gyratory discharge	2	1,500	3,000
Total	10		24,800

Underground Crushing, Exhaust System No. 7

Hood Location	Hoods, No.	Volume in CFM	
		Each	Total
Receiving bins (present)	1		20,590
Receiving bins (proposed)	1		16,000
Gyratory crusher enclosures (proposed)	2	2,500	5,000
Vibrating grizzly feed enclosures (proposed)	2	2,500	5,000
Vibrating grizzly undersize chutes (proposed)	2	2,000	4,000
Total (proposed)	7		30,000

dary exhaust from above and below eliminates dust dissemination at these locations.

The major portion of the air is taken from tight-fitting housings over the conveyors receiving the screened and crushed product. Air is exhausted from the head and tail of each conveyor housing with exhaust hoods several feet from and well above feed points on the conveyors.

A total of 22,000 cfm is exhausted from the screen and conveyor enclosures by a type W Rotoclone, with volumes exhausted at each respective hood as listed in Table II.

The total measured volume slightly exceeds that of calculated design, but all branch volumes fall well within the 10 pct deviation, which is the goal set by Kennecott's ventilation design department. If at all possible, as shown in Fig. 7, hoods are located on the housing or over a conveyor transfer point so that a vertical riser precedes a horizontal run, thus minimizing the possibility of material being projected into the branch ducting.

Secondary Screening Towers: The Symons crushers product and the undersize from the primary screens are conveyed to a 360-ton capacity surge bin, situated immediately above the secondary screening plant. The ore from the bin is choke fed to 10 screens. The undersize, screened to about 2 pct on $\frac{3}{4}$ -mesh screens, is conveyed directly to the shipping bins. The oversize material, including the underground ore, after passing through two gyratory crushers, is in closed circuit between the roll crushers and the secondary screens.

As illustrated in Fig. 8, the screens, as well as the oversize discharge chutes, are completely enclosed. The screen enclosures are ventilated indirectly by secondary exhaust from both the oversize and undersize product conveyor housings. The conveyors carrying the oversize from the screens are housed for the entire length of the screening tower with exhaust points at each end. The undersize product conveyor is exhausted at the head and tail with exhaust points between each screen undersize chute as indicated in Fig. 9. Through past experience this was found necessary because of the ore fall acting as a damper to air flow, which reduced the control of dust dispersed by the piston-like action of the ore as it strikes the conveyor. The exhaust points are out of line of direct ore fall, re-



Fig. 9—Secondary screen undersize product conveyor enclosure, showing exhaust between each ore drop.

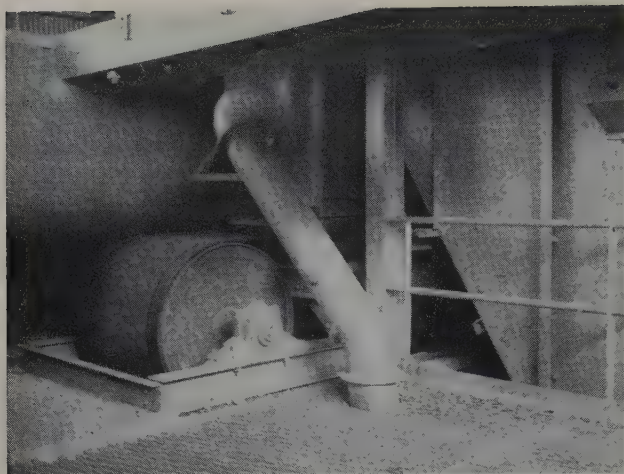


Fig. 10—Exhaust at tail of secondary screen oversize product conveyor. Note clean-up chute at end of housing and method of lighting ore chutes.

cessed, and connected to the enclosure about 5 ft above the conveyor. Fig. 10 shows an exhaust point at the tail of one of the screen oversize product conveyor enclosures. Also shown is a typical clean-up chute provided with a self-closing door. Sufficient volume is exhausted from the surge bin to maintain a negative pressure in the bin at all times. The air is exhausted from the top of the enclosed bin toward both ends. Air volumes exhausted from each hood can be found in Table II.

A volume of 22,500 cfm is exhausted by a type N Rotoclone, used as a collector-exhauster. Constant dribble from the conveyors leading to and from the screening tower made it necessary to house the take-up pulleys and provide dribble trays under the conveyors. Belt scrapers have been found to be of little value, especially when the ore is damp.

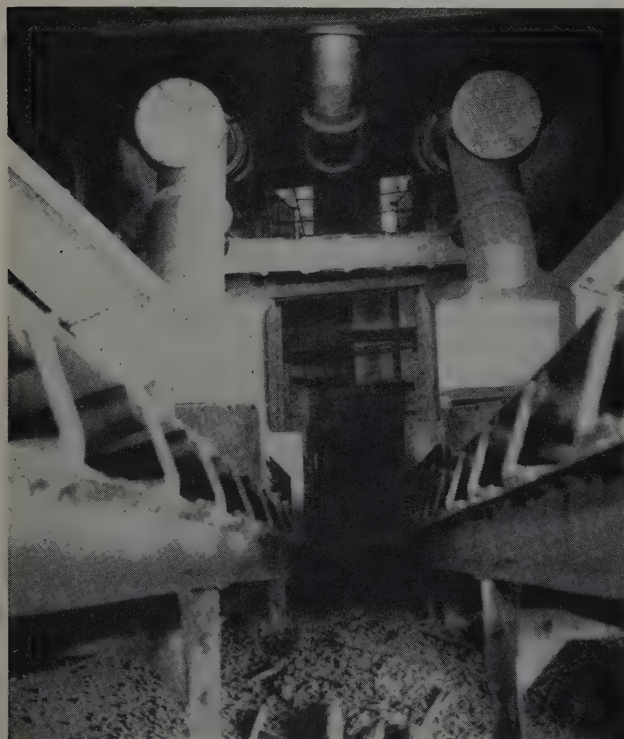


Fig. 11—Gyratory and roll crusher product feed enclosures, ducting and hoods partly shown.

Where possible the material that sluffs off the belts is discharged by chute to an exhausted enclosure.

Rolls and Gyratory Crushers: The two 72-in. roll crushers are in closed circuit with the secondary screens, handling the screen oversize which is a mixture of both pit and underground ore. Two No. 8 Gates gyratory crushers reduce the oversize material passing over the vibrating grizzlies to about a 2-in. product. Grizzly undersize material is returned to the surge bin along with that of the products from the gyratory and roll crushers.

Like the other crusher discharges, the feed points on the conveyor are completely housed, including a short section of conveyor between the gyratory and roll crushers, see Fig. 11. Openings between the enclosure and the vibrating machines are closed with old conveyor belting to reduce inordinate air inspiration.

One exhaust system using a type N Rotoclone collector-exhauster pulls 24,800 cfm from 10 hoods, 4 on each conveyor enclosure and 1 on each roll crusher feed box. The conveyor enclosure exhaust points are located at the front and rear of each ore drop.

At the present time the underground ore-receiving bins are ventilated, but no ventilation exists over the vibrating grizzlies or gyratory crushers. Designs are in the preliminary stage for additional ventilation in this area. A total of 30,000 cfm will be exhausted by existing equipment.

Housekeeping

A central vacuum cleaning system has been ordered to facilitate clean-up work in the secondary crushing plant. The unit is designed to handle two —2-in. or four 1½-in. hoses continuously. The network of piping extends to every nook and corner

Table III. Sample Mill Exhaust System — System No. 8

Hood Location	Hoods, No.	Volume in CFM
No. 1 transfer	1	3,000
No. 2 transfer, head pulley	1	1,500
No. 2 transfer, drop point	1	3,500
Sample pan-feeder	1	700
Rolls feed chute	1	1,000
Elevator discharge	1	700
Splitter enclosure	1	1,600
Total	7	12,000

in the secondary as well as the conveyor way leading to the storage bins. Long radius drainage fittings with self-closing valves will be used exclusively. Material collected by the primary and secondary separators will be emptied on the final products conveyor. At least for the present, no vacuum cleaning system has been planned for the primary crushing plant. Clean-up chutes with self-closing doors have been provided at the tail of all conveyors, eliminating air losses through the doors often left open after the clean-up work is done.

Sample Mill

A sample of ore is cut periodically at the final product conveyor transfer point adjacent to the sample tower. The sample is crushed by 24-in. rolls, elevated by a bucket elevator, and a portion selected by a rotary splitter. The splitter box, rolls crusher, elevator housing, and pan-feeder are furnished with exhaust ventilation.

Included in the same system is the exhaust from two conveyor transfer points, one as the ore leaves

the secondary crushing plant and one in proximity to the sample tower. At the sample tower conveyor transfer the head pulley enclosure is exhausted as well as the ore drop, because the cutting of the sample creates an abnormal disturbance that cannot be overcome by the exhaust point lower down. A type N Rotoclone exhausts 12,000 cfm from 7 hoods as shown in Table III.

Table IV lists the collecting equipment and characteristics pertaining to each of the eight systems described.

Table IV. Collector and System Specifications for Crushing Operations Exhaust Systems

System Number	Collector	Type	Size	Total Pressure, In., W.G.	HP Required
1	Rotoclone	W	36	2.5	66.8
2	Rotoclone	W	30	2.9	44.0
3	Rotoclone	W	16	1.8	9.8
4	Rotoclone	W	30	3.3	58.2
5	Rotoclone	N	32	8.0*	40.0
6	Rotoclone	N	32	8.2*	47.0
7	Spray Chamber			Approx. 4.7*	Not Determined
8	Rotoclone	N	16	7.0*	21.6

* System loss calculated as static pressure rather than total pressure.

Since ore storage bins superstructure is open to atmosphere on all sides, there is little need for ventilation. It is only for short periods that the one man employed must enter the dusty areas, during which time he wears a dust respirator.

Plant Atmospheric Conditions

Quarterly air sampling surveys are made in all departments by the Southwest field engineer. At these times dust counts are taken near crushing and screening operations. Since the secondary crushing plant has been in operation for only a short time, complete data is lacking, but Table V gives an account of dust concentrations as sampled to date.

Table V. Dust Concentrations in Crushing Plants and Sample Mill

Sample Location	Sam- ples, No.	Dust Concentration Million Part. Per Cu Ft		
		High	Low	Average
Primary Crushing Plant				
Crusher operating floor	4	4.0	0.8	1.8
Crusher drive floor	7	6.4	0.8	3.3
Conveyor tunnel	5	4.8	1.6	4.0
Secondary Crushing Plant				
Symons crushers operating floor	4	4.0	1.6	2.2
Symons crushers drive floor	3	2.4	1.6	1.9
Screen tower operating floor	5	5.6	0.8	2.7
Screen tower feed floor	3	1.6	0.8	1.1
Tunnel under screens	5	8.0	1.6	3.8
Roll mill operating floor	3	5.6	0.8	2.9
Roll mill feed floor	2	8.0	1.6	4.8
Conveyor tunnel under rolls	1	7.2	7.2	7.2
Sample Mill				
Ground floor	1	1.6	1.6	1.6
Sample cutter floor	2	1.6	0.8	1.2
Junction house	1	0.8	0.8	0.8

The dust counts were taken under normal operating conditions with an M.S.A. Midget impinger, counted on Hatch cells, with a micro-projector, according to the light field technique. Even the high counts are within the 10 million particles per cu ft of air, designated as the maximum allowable concentration for silica bearing dusts, according to the Arizona code.

The proximity of the hospital and housing projects to the crushing plants was one of the deciding factors in selecting the type of dust-collecting equipment. The effluent from the stacks was to be kept to a minimum and the idea of placing collected dust back in the ore circuit as dry material was rejected. Consequently, because of its high efficiency rating, a dynamic precipitator type of collector was chosen.

Type W Rotoclones were used on the first four systems installed. However, because water at Ray is at a premium, the last three systems in the secondary crushing plant were equipped with type N Rotoclones. Soluble copper salts, present in varying amounts in the mine ore, have created a corrosion problem in the collectors. By nature of the construction of the type N, these collectors have shown the greatest wear. It was also found necessary to apply a protective coating to all inside parts of the Rotoclones, except stainless steel parts.

If the water problem at Ray becomes more acute, some method for reclaiming the dust collector water may have to be employed. However, this is not anticipated for the near future.

A Changing Trend

Like the old sweatshop, dirty mines, mills, and crushing plants are gradually becoming a thing of the past. Management throughout the country is aware that dust, mist, and fume control contribute appreciably to better industrial relations and minimize labor turnover. Evaluating ventilation and air conditioning is somewhat more difficult than measuring metallurgical gains or losses; however, in many cases, the cost of labor turnover attributable to unsatisfactory environmental conditions will amortize ventilation systems.

It cannot be too emphatically expressed that ventilation installations are much less expensive if incorporated with the original design of the plant. Superimposing is not only more expensive as a rule, but often limits the engineer in providing adequate protection.

Notwithstanding a well-engineered transporting system, the proper selection of fans and collecting equipment may be the secret of success or failure of many types of exhaust ventilation.

Acknowledgements

The author wishes to thank Louis Buchman, Vice President and General Manager, Western Operations, and A. P. Morris, Ray Mines Division General Manager, for their permission to write this paper. He expresses appreciation to C. L. Hoyt and Otis W. Boise for preparing data used in the paper, to Paul Whitehead for obtaining photographs, and to H. W. McCullough for typing the manuscript.

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Cleaning Various Coals In A Drum-Type Dense-Medium Pilot Plant

by M. R. Geer, W. A. Olds, and H. F. Yancey

THE increase in the number of coal-cleaning plants employing dense-medium processes occurring since 1946 is especially interesting when viewed historically. Both sand and magnetite were introduced as material for heavy mediums at about the same time, sand in the Chance process in 1921 and magnetite in the Conklin process in 1922, but from that point on their records diverge. The Chance process enjoyed a steady growth from its inception, whereas no additional magnetite plants were built in the United States for over 20 years. Then, following the close of the World War II, magnetite was again introduced, this time with marked success. During the following years some 47 plants employing magnetite medium were built.

This rapid growth of dense-medium cleaning has been concurrent with widespread adoption of full-seam mining on one hand and a return to a more competitive market on the other. At a time when changing mining practice has provided cleaning plants with dirtier coal and changing market conditions have simultaneously demanded a cleaner product, the industry has through necessity turned to improved preparation. The inherently greater sharpness with which dense-medium processes can separate coal from impurity is thus helping to hold the line against ever-increasing mining costs and at the same time assisting materially in retaining badly needed markets.

Although dense-medium cleaning unquestionably offers a distinct advantage when the washing problem is difficult, other methods can provide almost equally high efficiency when the coal is easy to wash. Moreover, fine coal cannot yet be treated by heavy medium in a proved process, although the Driessen cyclone is in the pilot-plant stage.

Most of the present types of dense-medium equipment have been in use only a few years, and the dearth of information in the literature concerning their performance characteristics is entirely understandable. Nevertheless this information is necessary if the process is to be intelligently applied to individual cleaning problems. Without data on the efficiency of a process in a particular type of separation, it is difficult to assess the advantage to be expected from it. Similarly, the role of particle size in heavy-medium separation is important in some cases, yet there is little published information on this aspect. To mention only one more of the numerous points on which essential information is lacking, the bearing of medium characteristics on performance has been discussed only in qualitative terms.

It was with the hope of providing information on some of these points that the Bureau of Mines built

a dense-medium pilot plant for cleaning coal at its Northwest Experiment Station in Seattle in 1950. The plant has been operated continuously since that time, and over 50 runs have been made on 7 coals exhibiting a wide range of washability characteristics. An idea of the magnitude of this work will be gained from the fact that examination of the plant products has involved some 600 float-and-sink separations and about 2500 ash determinations.

A laboratory pilot plant is especially well adapted to investigate many aspects of performance because close control over test conditions can be exercised and because a large number of tests can be made rapidly. On the other hand, factors such as consumption of medium and other cost items can be investigated satisfactorily only in a commercial plant. Actually, the two forms of investigation should be complementary, with the laboratory work pointing the way for confirming tests in commercial units.

Pilot Plant

The dense-medium pilot plant employed for this work comprises a 24x30-in. drum-type separating vessel, a 12-in. densifier, a 12-in. magnetic separator, a 26-in. x 9-ft vibrating screen, and the necessary pumps and conveyors for handling materials. Arrangement of these units corresponds with the flowsheet used in most commercial plants, except that a thickener is not provided for the feed to the magnetic separator. Coal and refuse discharge from the separating drum to the vibrator, which is divided longitudinally down the center. Medium draining through the first 3 ft of the screen is recirculated directly to the drum. Sprays on the middle 3 ft of the screen rinse medium from the products, and the last 3-ft section is for dewatering. Dilute medium from the rinsing and dewatering sections is pumped to the magnetic separator, where magnetic solids are recovered. These are pumped to the densifier, from which they return to the medium-drainage sump by gravity through a demagnetizing coil.

The drum-type separating vessel is a scale model of a commercial unit. Feed enters axially at one end of the drum just below the surface of the bath, and float material overflows through a circular opening at the other end. Particles sinking to the bottom of the bath are picked up by lifting flights bolted to the inner wall of the drum, elevated out of the bath, and sluiced to the vibrating screen. Baffles suspended in the bath prevent float material from entering the sink-lifting flights.

About 8 gpm of medium is used to sluice the feed into the drum. An additional 15 to 24 gpm, depending upon operating conditions, is added through two pipes dipping into the bath behind the baffle on the side where the sink-lifting flights enter the bath.

The bath available for separation is 2 ft long and 13 in. wide, giving an area of 2.08 sq ft. Depth of bath from the surface to the top of the sink-lifting flights, measured vertically below the axis, is 6 in.

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Discussion on this paper, TP 3551F, may be sent (2 copies) to AIME before Sept. 30, 1953. Manuscript, Jan. 19, 1953. Los Angeles Meeting, February 1953.

Table I. Identification of Coals Tested

State	County	Mine	Bed
Alaska		Buffalo	No. 2
Alaska		Buffalo	No. 5
Montana	Cascade	Surmi	Belt Creek
Washington	King	McKay Strip	Raven
Washington	Pierce	Skookum	No. 3
Washington	Whatcom	Bellingham	Bellingham
Wyoming	Carbon	Nugget	No. 1, bottom bench

when the head over the weir is $1\frac{1}{8}$ in. Throughputs ranging from 1.5 to 5.2 tons per hr were used in the test work, but most runs were made with a feed rate of about 1.75 tons per hr. Thus the average capacity was about 0.85 tons per sq ft per hr, or from 20 to 30 pct of that attained in commercial units of the same type. The lower throughput per unit area is necessary to provide a separating time comparable to that in a commercial vessel. A particle of float travels the 2-ft length of the small drum in 13 sec, approximately equalling the time allowed for the 6-ft length of a commercial vessel of the same type. With the same treatment time, the commercial drum obtains about three times the throughput per unit area by employing a horizontal current three times as great.

In comparing drums of such dissimilar size the ratio of coal to medium is more indicative of bed crowding than throughput per unit area. The small drum treats 2.5 to 6.7 lb of feed per gal of medium, while the large size treats 4 to 6. Thus, from the standpoint of both the time available for separation and the degree of crowding in the bath where separation takes place, the laboratory unit appears to be comparable to its commercial-size counterpart.

The most serious shortcoming of the small drum is its inability to deal with the full range of sizes normally treated in a commercial-size unit. Owing largely to the shallowness of the bath, and the comparatively low head of medium over the discharge weir required to insure adequate treatment time, coal coarser than 2 in. square hole size cannot be handled. All but a few of the results presented in this report were obtained on feeds of 2 to $\frac{1}{4}$ in., square hole; in the remaining tests, the bottom size was $\frac{1}{8}$ in., square hole.

Test Procedure

In most commercial plants, where the same coal or the same mixture of coals always is treated, the character of the medium required for satisfactory plant operation is not subject to great change. Moreover, the character of the medium used is likely to be dictated jointly by the nature of the impurities in the raw coal and the operating characteristics of the plant. In the pilot plant, however, where coals of widely different nature are cleaned, and separations ranging from 1.30 to 1.75 sp gr are required, the character of the medium is an all-important variable. Therefore the first step in preparing the pilot plant for a new operation was to select, and make provision to maintain, suitable medium.

The medium characteristic of principal operating importance is stability. If the medium is too unstable, either because the magnetite is excessively coarse or because the coal being treated supplies no clay to assist in stabilization, the differential in density between the top and bottom of the bath is undesirably high. Conversely, if the medium is too

stable, either because of excessively fine magnetite or the presence of an undue amount of clay, the small density differential generally desired is not obtained. For separations at 1.55 sp gr or higher, commercial grade A magnetite, 60 to 75 pct through 325-mesh, proved satisfactory, although with some coals containing large amounts of clay a high medium-cleaning rate was needed to preserve the desired density differential in the bath. Grade B magnetite, 90 pct through 325 mesh, was required in treating coals free of clay at a sp gr of 1.40 or less. In the range between 1.40 and 1.55, either mixtures of grade A and B were used or the coarser fractions of grade A were removed from the medium by classification to obtain the desired stability. The subjects of medium stability and medium contamination are dealt with more completely elsewhere in the report.

With suitable medium circulating in the plant, coal prepared to the desired size is fed to the drum. If sufficient coal is available, up to 10 tons is run through the plant for a test, giving an operating period of about 5 hr. Often, however, the number of tests under different operating conditions desired for a coal necessarily limits the duration of the individual tests to about a half hour each. Regardless of the duration of the run, all of the washed coal and refuse is weighed to determine yield and is sampled for subsequent examination.

The washed coal and refuse samples were subjected to float-and-sink tests at each 0.1 interval of sp gr from 1.30 to 1.80. Each individual density

Table II. Specific-gravity Analyses of Coals Tested

Coal	Sp Gr	Wt, Pct	Ash, Pct	Cumulative	
				Wt, Pct	Ash, Pct
Buffalo No. 2	Under 1.30	58.9	2.2	58.9	2.2
	1.30 to 1.40	21.9	8.7	80.8	4.0
	1.40 to 1.50	6.0	22.9	86.8	5.3
	1.50 to 1.60	4.3	32.5	91.1	6.6
	1.60 to 1.70	2.2	38.8	93.3	7.3
	Over 1.70	6.7	61.9	100.0	11.0
Buffalo No. 5	Under 1.30	49.4	1.7	49.4	1.7
	1.30 to 1.40	13.6	10.9	63.0	3.7
	1.40 to 1.50	5.8	24.1	68.8	5.4
	1.50 to 1.60	4.2	33.1	73.0	7.0
	1.60 to 1.70	3.9	42.1	76.9	8.8
	1.70 to 1.80	3.0	49.1	79.9	10.3
	Over 1.80	20.1	69.6	100.0	22.2
Surmi	Under 1.30	1.2	2.6	1.2	2.6
	1.30 to 1.40	37.6	8.4	38.8	8.2
	1.40 to 1.50	23.9	18.0	62.7	11.9
	1.50 to 1.60	11.0	26.8	73.7	14.2
	1.60 to 1.70	6.2	34.6	79.9	15.8
	1.70 to 1.80	3.9	42.0	83.8	17.0
	Over 1.80	16.2	59.0	100.0	23.8
Raven	Under 1.30	6.1	3.7	6.1	3.7
	1.30 to 1.40	47.3	13.0	53.4	11.9
	1.40 to 1.50	21.0	25.4	74.4	15.7
	1.50 to 1.60	5.4	37.1	79.8	17.2
	1.60 to 1.70	6.1	47.2	85.9	19.3
	1.70 to 1.80	5.2	57.8	91.1	21.5
	Over 1.80	8.9	73.1	100.0	26.1
Skookum	Under 1.30	9.8	5.2	9.8	5.2
	1.30 to 1.40	28.0	13.9	37.8	11.6
	1.40 to 1.50	17.1	24.7	54.9	15.7
	1.50 to 1.60	6.5	34.1	61.4	17.7
	1.60 to 1.70	3.8	42.9	65.2	19.1
	1.70 to 1.80	3.0	50.1	68.2	20.5
	Over 1.80	31.8	74.4	100.0	37.6
Bellingham	Under 1.30	0.3	3.7	0.3	3.7
	1.30 to 1.40	53.6	10.9	53.9	10.9
	1.40 to 1.50	22.3	21.4	76.2	13.9
	1.50 to 1.60	8.1	33.1	84.3	15.8
	1.60 to 1.70	4.2	43.1	88.5	17.1
	1.70 to 1.80	2.6	51.5	91.1	18.1
	Over 1.80	8.9	72.4	100.0	22.9
Nugget	Under 1.30	14.3	5.2	14.3	5.2
	1.30 to 1.40	60.3	10.0	74.6	9.1
	1.40 to 1.50	8.9	24.4	83.5	10.7
	1.50 to 1.60	5.5	35.9	89.0	12.3
	1.60 to 1.70	3.6	45.9	92.6	13.6
	Over 1.70	7.4	66.1	100.0	17.5

Table III. Specific-gravity Analyses of Individual Sizes of Bellingham Washed Coal

Size, In.	Sp Gr	Wt, Pct	Ash, Pct	Cumulative	
				Wt, Pct	Ash, Pct
2 to 1 Wt, 23.3 pct	Under 1.30	0.0		0.0	
	1.30 to 1.40	64.5	11.6	64.5	11.6
	1.40 to 1.50	32.2	21.6	96.7	14.9
	1.50 to 1.60	3.3	29.3	100.0	15.4
	1.60 to 1.70	0.0		100.0	15.4
	1.70 to 1.80	0.0		100.0	15.4
1 to ½ Wt, 41.8 pct	Under 1.30	0.2	4.7	0.2	4.7
	1.30 to 1.40	72.7	11.1	72.9	11.1
	1.40 to 1.50	23.7	21.4	96.6	13.6
	1.50 to 1.60	3.4	30.8	100.0	14.2
	1.60 to 1.70	0.0		100.0	14.2
	1.70 to 1.80	0.0		100.0	14.2
½ to ¼ Wt, 24.3 pct	Under 1.30	0.7	3.3	0.7	3.3
	1.30 to 1.40	58.0	10.3	58.7	10.2
	1.40 to 1.50	33.2	21.7	91.9	14.4
	1.50 to 1.60	7.7	32.1	99.6	15.7
	1.60 to 1.70	0.4	40.3	100.0	15.8
	1.70 to 1.80	0.0		100.0	15.8
¼ to 0 Wt, 10.6 pct	Under 1.30	0.6	3.3	0.6	3.3
	1.30 to 1.40	70.2	9.4	70.8	9.3
	1.40 to 1.50	21.6	20.1	92.4	11.9
	1.50 to 1.60	6.3	30.9	98.7	13.1
	1.60 to 1.70	0.9	40.3	99.6	13.3
	1.70 to 1.80	0.2	45.0	99.8	13.4
Composite Wt, 100.0 pct	Under 1.30	0.3	3.7	0.3	3.7
	1.30 to 1.40	66.9	10.9	67.2	10.9
	1.40 to 1.50	27.9	21.4	95.1	14.0
	1.50 to 1.60	4.7	31.1	99.8	14.8
	1.60 to 1.70	0.2	40.3	100.0	14.8
	1.70 to 1.80	0.0		100.0	14.8

fraction obtained in this way was then screened at 1, ½, and ¼ in. to provide information on the character of the separation effected in the individual sizes of the feed. Ash analyses were made on all size-density fractions.

Coals Tested

Table I identifies coals treated in the pilot plant. Table II shows specific-gravity analyses.

The two Alaska coals are from the Buffalo mine in the Matanuska field. The coal from the No. 2 bed is the cleanest of any tested. In raw form it contains only 11.0 pct ash and 6.7 pct of impurity heavier than 1.70 sp gr. The impurity is largely shale and sandy shale, neither of which disintegrates appreciably in water or is noticeably tabular in shape. The Buffalo No. 5 coal contains over twice as much ash and 20.1 pct impurity heavier than 1.80 sp gr. A substantial part of this impurity is carbonaceous shale, the characteristic particle shape of which was tabular. Both Buffalo coals are relatively easy to wash. The cleanest portion of both coals is lighter than 1.30 sp gr, neither contains an unusually large proportion of material of intermediate density, and even in the No. 5 coal the percentage of heavy impurity is not unduly high.

The Surmi coal, from the Belt Creek bed in Cascade County, Mont., is somewhat more difficult to clean than the Buffalo No. 5. It contains about twice as much material in the range from 1.50 to 1.70 sp gr, and only 1 pct is lighter than 1.30 sp gr. The impurity heavier than 1.80 sp gr, amounting to 16.2 pct, is largely heavy bone. In fact, this coal is singularly free of shale, and contains no clay. The bone does not differ from the coal in particle shape.

The Raven coal, which originated from a strip mine in King County, Wash., is much like Surmi in that the clean coal bulks largely in the range from 1.30 to 1.50 sp gr. The 8.9 pct impurity heavier than 1.80 sp gr present is not in itself formidable. How-

ever, this heavy impurity is almost entirely shale and heavy bone, both of which have a distinctly tabular particle shape that renders them difficult to separate.

The Skookum coal, from Pierce County, Wash., is unquestionably the most difficult coal of the group to clean. The clean coal in this bed is much more friable than the shales with which it is associated and therefore concentrates in the sizes smaller than ½ in., which were not treated by heavy medium. The feed for the heavy-medium tests analyzed 37.6 pct ash, and contained 31.8 pct impurity heavier than 1.80 sp gr. This amount of impurity in itself constitutes a washing problem of more than usual difficulty. In addition, the coal contains more material of intermediate density than usual, and the proportion of clean coal lighter than 1.30 sp gr, 9.8 pct, is low.

The Bellingham coal, from Whatcom County, Wash., contains almost no material lighter than 1.30 sp gr. Nearly 75 pct is in the range from 1.30 to 1.50. The 8.9 pct impurity heavier than 1.80 sp gr is largely clay that disintegrates readily in water, thus adding somewhat to the difficulty of cleaning.

The Nugget coal is from the bottom bench of the No. 1 bed in a strip mine located at Hanna, Wyo. It contains only 7.4 pct impurity heavier than 1.70 sp gr, largely bone and carbonaceous shale. Sixty percent of the coal occurs in the range 1.30 to 1.40 sp gr, an additional 14 pct being lighter than 1.30.

Evaluation of Performance

Dividing all washed coal and refuse samples into seven specific-gravity fractions, subdividing each of these fractions into four size groups, and finally analyzing every size-density component for ash content is a time-consuming, costly procedure, but nothing short of this method provides the complete, basic data essential in evaluating the performance of any cleaning unit. Naturally space limitations

Table IV. Specific-gravity Analyses of Individual Sizes of Bellingham Refuse

Size, In.	Sp Gr	Wt, Pct	Ash, Pct	Cumulative	
				Wt, Pct	Ash, Pct
2 to 1 Wt, 18.9 pct	Under 1.30	0.0		0.0	
	1.30 to 1.40	0.0		0.0	
	1.40 to 1.50	1.3	23.7	1.3	23.7
	1.50 to 1.60	39.9	34.5	41.2	34.2
	1.60 to 1.70	23.1	43.4	64.3	37.5
	1.70 to 1.80	11.5	51.2	75.8	39.6
1 to ½ Wt, 32.3 pct	Under 1.30	0.0		0.0	
	1.30 to 1.40	0.0		0.0	
	1.40 to 1.50	0.7	24.6	0.7	24.6
	1.50 to 1.60	27.9	35.3	28.6	35.0
	1.60 to 1.70	23.8	43.4	52.4	38.8
	1.70 to 1.80	13.8	52.2	66.2	41.6
½ to ¼ Wt, 26.3 pct	Under 1.30	0.0		0.0	
	1.30 to 1.40	0.1	10.1	0.1	10.1
	1.40 to 1.50	0.3	23.6	0.4	20.2
	1.50 to 1.60	13.6	35.3	14.0	34.9
	1.60 to 1.70	23.6	43.2	37.6	40.1
	1.70 to 1.80	17.1	51.6	54.7	43.7
¼ to 0 Wt, 22.5 pct	Under 1.30	0.0		0.0	
	1.30 to 1.40	0.3	7.7	0.3	7.7
	1.40 to 1.50	0.4	21.6	0.7	15.6
	1.50 to 1.60	5.2	32.9	5.9	30.9
	1.60 to 1.70	9.6	41.8	15.5	37.6
	1.70 to 1.80	8.6	50.5	24.1	42.2
Composite Wt, 100.0 pct	Under 1.30	0.0		0.0	
	1.30 to 1.40	0.1	8.9	0.1	8.9
	1.40 to 1.50	0.6	23.7	0.7	21.5
	1.50 to 1.60	21.3	34.9	22.0	34.5
	1.60 to 1.70	20.4	43.2	42.4	38.7
	1.70 to 1.80	13.1	51.6	55.5	41.8

Table V. Specific-gravity Analyses of Individual Sizes of Bellingham Feed

Size, In.	Sp Gr	Wt, Pct	Ash, Pct	Cumulative	
				Wt, Pct	Ash, Pct
2 to 1	Under 1.30				
Wt, 22.4 pct	1.30 to 1.40	53.6	11.6	53.6	11.6
	1.40 to 1.50	27.0	21.6	80.6	14.9
	1.50 to 1.60	9.5	33.0	90.1	16.9
	1.60 to 1.70	3.9	43.4	94.0	18.0
	1.70 to 1.80	1.9	51.2	95.9	18.6
	Over 1.80	4.1	63.8	100.0	20.5
1 to ½	Under 1.30	0.2	4.7	0.2	4.7
Wt, 39.9 pct	1.30 to 1.40	60.9	11.1	61.1	11.1
	1.40 to 1.50	20.0	21.4	81.1	13.6
	1.50 to 1.60	7.4	33.6	88.5	15.3
	1.60 to 1.70	3.8	43.4	92.3	16.5
	1.70 to 1.80	2.2	52.2	94.5	17.3
	Over 1.80	5.5	68.2	100.0	20.1
½ to ¼	Under 1.30	0.6	3.3	0.6	3.3
Wt, 24.7 pct	1.30 to 1.40	45.7	10.3	46.3	10.2
	1.40 to 1.50	26.2	21.7	72.5	14.4
	1.50 to 1.60	9.0	33.1	81.5	16.4
	1.60 to 1.70	5.3	43.0	86.8	18.1
	1.70 to 1.80	3.6	51.6	90.4	19.4
	Over 1.80	9.6	70.3	100.0	24.3
¼ to 0	Under 1.30	0.4	3.3	0.4	3.3
Wt, 13.0 pct	1.30 to 1.40	45.9	9.4	46.3	9.3
	1.40 to 1.50	14.2	20.1	60.5	11.9
	1.50 to 1.60	5.9	31.5	66.4	13.6
	1.60 to 1.70	3.9	41.6	70.3	15.2
	1.70 to 1.80	3.2	50.2	73.5	16.7
	Over 1.80	26.5	78.8	100.0	33.2
Composite	Under 1.30	0.3	3.7	0.3	3.7
Wt, 100.0 pct	1.30 to 1.40	53.6	10.9	53.9	10.9
	1.40 to 1.50	22.3	21.4	76.2	13.9
	1.50 to 1.60	8.1	33.1	84.3	15.8
	1.60 to 1.70	4.2	43.1	88.5	17.1
	1.70 to 1.80	2.6	51.5	91.1	18.1
	Over 1.80	8.9	72.4	100.0	22.9

require that such voluminous data be presented in only summary form in a report of this kind, but an example of the complete information for a test in the pilot plant is presented in this section to illustrate the methods of interpreting and summarizing data employed in the balance of the report.

Tables III and IV, respectively, show the specific-gravity analyses of the individual-size fractions of washed coal and refuse produced in a test on Bellingham coal. Table V shows comparable data for the feed, calculated from the analyses of the two products; such calculated feeds were used exclusively in the present work to eliminate as completely as possible the influence of degradation.

The overall result of this test was to produce a washed coal of 14.8 pct ash from a feed of 22.9 pct, with a yield of 80.0 pct; the 20.0 pct refuse rejected analyzed 55.4 pct ash. Examination of the specific-gravity analyses of washed coal shown in Table III shows that the 2 to 1-in. and 1 to ½-in. sizes were entirely free of material heavier than 1.60 sp gr. The ½ to ¼-in. size contained 0.4 pct of such material, but its presence increased the ash content of that size by only 0.1 pct and had no effect on the ash content of the washed coal as a whole. Only in the degradation material finer than ¼ in. was there any impurity heavier than 1.70 sp gr.

Similarly, examination of the data for refuse in Table IV discloses that only in the coarsest size group was there more than 1 pct of coal lighter than 1.50 sp gr. Considering all sizes together, less than 0.5 pct of the refuse was coal analyzing 14.8 pct ash, the quality of the washed coal produced.

Table VI summarizes the ash reduction effected in each size of the raw coal, and shows the corresponding Fraser and Yancey efficiency values. Efficiency, as the term is used in this report, is the ratio, expressed in percentage, of the yield of washed coal actually obtained to the yield of float coal of the

same ash content shown to be present in the feed by specific-gravity analysis. The efficiency for all sizes combined was 99.9 pct, while that for the ½ to ¼-in. fraction was 99.5 pct. The value of 98.6 pct for the ¼-in. to 0 material is not particularly significant because this size is formed largely through degradation, much of which takes place after the coal leaves the separating bath.

Also presented in Table VI are the distribution data for the test, which shows what percentage of each density fraction of the feed was recovered in the washed coal. These data are calculated from the specific-gravity analyses of the washed products. In the two coarsest size fractions there was perfect recovery of all coal lighter than 1.40 sp gr, and perfect elimination of all material heavier than 1.60. In the ½ to ¼-in. fraction recovery of coal was almost perfect, but elimination of impurity was impaired slightly; 6.1 pct of the 1.60 to 1.70 sp gr fraction entered the washed coal. The influence of particle size in the heavy-medium bath is shown most clearly in the treatment of material of intermediate density. In the 1.50 to 1.60 sp-gr fraction, for example, the percentage recovery increased progressively with decrease in particle size, from 29.1 in the 2 to 1-in. group to 67.6 in the material of ½ to ¼-in.

Fig. 2 shows the distribution curve obtained by plotting the distribution data for the combined sizes of feed. The method of plotting this curve and its significance has been discussed previously.¹ Throughout the present report the specific gravity of separation is that at which the distribution curve crosses the 50 pct ordinate, indicating that material of this specific gravity was divided equally between washed coal and refuse. As shown by the curve, the specific gravity of separation for all sizes combined was 1.55. The two coarsest fractions of the feed were separated at 1.54, while the ½ to ¼-in. material was separated at 1.57.

The distribution curve also provides a convenient means for characterizing the sharpness with which

Table VI. Summary of Performance of Pilot Plant on Bellingham Coal

Item	Size Fraction, In.					Composite
	2 to 1	1 to ½	½ to ¼	¼ to 0	2 to 0	
Screen analysis of feed, pct	22.4	39.9	24.7	13.0	100.0	
Screen analysis of washed coal, pct	23.3	41.8	24.3	10.6	100.0	
Screen analysis of refuse, pct	18.9	32.3	26.3	22.5	100.0	
Ash content of feed, pct	20.5	20.1	24.3	33.2	22.9	
Ash content of washed coal, pct	15.4	14.2	15.8	13.5	14.8	
Ash content of refuse, pct	45.4	50.6	55.7	70.1	55.4	
Yield of washed coal, pct	83.2	83.8	78.7	65.2	80.0	
Theoretical yield, pct*	83.3	83.8	79.1	66.1	80.1	
Recovery efficiency, pct	99.9	100.0	99.5	98.6	99.9	
Distribution data, pct recovered in washed coal						
Under 1.30	100.0	100.0	100.0	100.0	100.0	
1.30 to 1.40	100.0	100.0	99.9	99.8	100.0	
1.40 to 1.50	99.2	99.4	99.7	98.9	99.5	
1.50 to 1.60	29.1	38.8	67.6	69.7	46.9	
1.60 to 1.70	0.0	0.0	6.1	15.7	3.8	
1.70 to 1.80	0.0	0.0	0.0	4.9	0.0	
Over 1.80	0.0	0.0	0.0	0.6	0.0	
Specific gravity of separation	1.54	1.54	1.57	1.59	1.55	
Error area	6	9	18		15	
Sink in washed coal, feed, pct†	1.2	1.2	1.6	1.1	1.2	
Float in refuse, feed, pct†	2.2	1.7	1.8	1.5	2.4	
Total misplaced material†	3.4	2.9	3.4	2.6	3.6	

* Float-and-sink yield at ash content of washed coal.

† At specific gravity of separation.

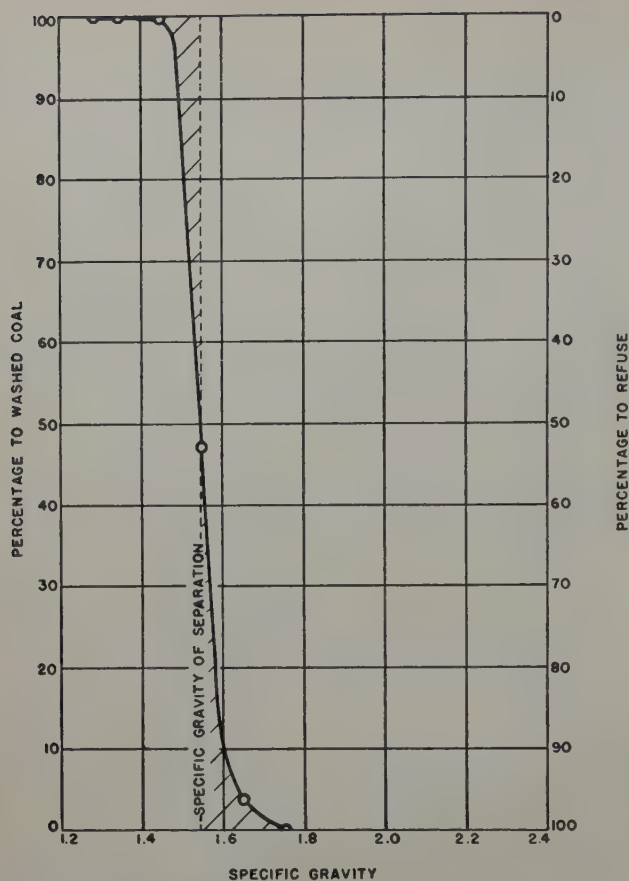


Fig. 1—Distribution curve for test on 2 to $\frac{1}{4}$ -in. Bellingham coal.

the separation between coal and impurity was made. The curve representing a perfect separation would be composed of three straight lines; at densities less than the specific gravity of separation this perfect curve would show 100 pct recovery in the washed product, and at higher densities it would indicate 100 pct rejection in the refuse product. The area between the distribution curve actually obtained and the curve representing a perfect separation is thus a measure of the sharpness of separation. This area, when the curve is plotted to a fixed scale, is termed *error area*. In Fig. 2 it is indicated by cross hatching. The error area for all sizes combined in this test was 15. By particle size, it varied from 6 in the 2 to 1-in. fraction to 18 in the material of $\frac{1}{2}$ to $\frac{1}{4}$ -in.

In a general way, error area is a characteristic of a particular cleaning device and is not influenced directly by the specific-gravity composition of the coal being cleaned. For a given device it varies through only a relatively narrow range. Recovery efficiency, on the other hand, is determined jointly by the characteristics of the cleaning device, the density at which the separation is made, and the specific-gravity composition of the raw coal. Thus it varies over a comparatively wider range, depending on the coals treated.

The percentages of misplaced material given in Table VI, sink in the washed coal and float in the refuse at the specific gravity of separation, are expressed in terms of feed. Like recovery efficiency, misplaced material varies with the character of the raw coal and the density of the separation.

Summary of Performance Data

Of the more than 50 test runs made in the pilot plant thus far, the results of 10 were selected for

presentation in Table VII to illustrate the full range of performance obtained. Six of these tests were made at specific gravities of 1.55 or higher, and four were made at 1.41 to 1.31; the results are thus divided into high and low-density groups.

In the high-density group, the results for the Nugget and Raven coals are clearly inferior, particularly as judged by error area and distribution data. The poor results obtained with these two coals are thought to be due to degradation, and will be discussed in detail under that heading. Considering the other four coals, however, the recovery efficiencies varied from 98.6 to 99.9 pct, and the corresponding spread in misplaced material was from 0.9 to 3.6 pct. Error areas ranged from 15 to 21. There was virtually complete recovery of material lighter than 1.50 sp gr in all coals, but the elimination of heavy impurity was less perfect. The percentage of sink 1.80 impurity entering the washed product ranged from nothing with Bellingham to 0.6 pct with Surmi. This imperfect rejection of impurity deserves explanation, especially as to why it varies with different coals.

With the Wilkeson No. 3 coal, the 0.2 pct of sink 1.80 material reporting to the washed product was all finer than $\frac{1}{8}$ in.; impurity coarser than that size was eliminated completely. Similarly, the impurity entering the washed coal in the Surmi and Buffalo tests was largely finer than $\frac{1}{4}$ in. The smaller sizes of impurity are particularly difficult to remove completely from the Surmi coal because they are composed almost entirely of bone rather than heavy shale. The impurity in the Buffalo coal is predominately flaky, carbonaceous shale that is harder to remove both because of particle shape and relatively low density. Thus, although particle size and shape are of less importance in dense-medium cleaning than in other methods, they still are factors of some influence.

The group of low-density tests recorded in Table VII were made at densities well below those employed in ordinary washery operation. They are thus of interest primarily in showing the performance to be expected in the clean-coal middling split of a three-product separation, or when skimming off a superclean coal at low density. The former of these applications is widely practiced in Europe today, and probably will become increasingly important in America. The latter may be required ultimately to provide special-purpose coal for chemical utilization processes such as hydrogenation.

The Buffalo No. 2 test, made at 1.30 sp gr, is a good example of using heavy-medium treatment to skim off the cleanest fraction of a coal. Washability data indicated that a fraction analyzing 2.5 pct ash was available at a theoretical yield of 61.0 pct. Operation of the plant provided a product of that quality at a yield of 57.3 pct. The recovery efficiency was thus 93.9 pct. Perfect elimination of all material heavier than 1.50 sp gr was obtained, but the recovery of coal lighter than 1.30 was only 89.4 pct. Such a low recovery of the lightest fraction was to be expected when operating at 1.30 sp gr, however, and as judged by the error area of 13, this test provided one of the sharpest separations accomplished in the pilot plant.

The test on Wilkeson No. 3 coal at 1.36 sp gr is of special interest because of the unusually low yield, 26.2 pct, of clean coal. Normally such low yields are encountered only in re-treating refuse material. It is noteworthy that although the elimination of mate-

rial heavier than 1.60 sp gr was perfect, and the recovery of coal lighter than 1.30 was 98.5 pct, the recovery efficiency was only 89.4 pct. An efficiency of this magnitude obtained in the treatment of an ordinary coal would represent extremely poor operation, but in the treatment of a material providing such a low yield of clean coal, it probably is as high as should be expected.

Lacking a direct correlation between the performance of the pilot plant and a commercial unit of the same type, and there has been no opportunity to make such a comparison, the probability of duplicating pilot-plant results in a full-size unit remains conjectural. Both the shallowness and shortness of the separating bath in the small drum would appear to militate against equaling the performance possible in a large bath. However, these limitations might be outweighed by other factors in operating a commercial plant, especially bed congestion or lack of good density control.

Indirect evidence that comparable performance is obtainable commercially is afforded by the results published by Hanson, Heinlein, and Vonfeld.² Whereas their drum separators are of a different type and bed congestion and retention time probably differ materially, their washing results are similar to those obtained in the pilot plant. Their distribution data for treatment of a 3 to 3/8-in. feed at 1.52 sp gr showed perfect recovery of all material lighter than 1.40, with 0.8 pct of the sink 1.80 impurity entering the washed coal. These figures correspond closely with average pilot-plant performance. Their data for a 6 to 3-in. feed showed perfect recovery of all coal lighter than 1.40 and perfect rejection of all impurity heavier than 1.60 sp gr. A separation of this sharpness was obtained in the pilot plant only with Bellingham coal. Naturally, a 6 to 3-in. feed should separate more sharply than one of 2 to 1/4-in.

Influence of Degradation on Performance Criteria

As pointed out in the discussion of Table VII, the distribution data for Raven and Nugget are clearly inferior to those obtained on the other coals, particularly from the standpoint of impurity rejection. As neither mechanical defects nor faulty operating

conditions existed to account for such poor performance, an explanation was sought in the character of the coals themselves. The refuse material in both coals was of normal density and ordinary particle shape; thus neither of these factors appeared to be responsible. The most reasonable answer seemed to be the influence of degradation of the washed products subsequent to the gravity separation.

In terms of rank both coals are on the line between high-volatile C bituminous and sub-bituminous A. When fresh they are more resistant to breakage than most bituminous coals, but on drying they become much more friable. If, in the handling incident to screening, sampling, and float-and-sink examination, substantial amounts of impurity were liberated from intergrown particles in the washed coal, the rejection of impurity by the plant would be made to appear poor. This hypothesis was not verified, however, until it was observed that stored Bellingham coal, high-volatile C bituminous, gave results in the plant that were distinctly poorer than those obtained on fresh coal. Here again the elimination of impurity appeared to be poor, and degradation was again blamed.

A testing procedure was devised to evaluate the influence of degradation on the stored Bellingham coal. About 1200 lb of the 2 to 1/4-in. coal was separated at 1.60 sp gr on a float-and-sink bath, using the same care exercised in the normal testing of samples. The float-and-sink products from this operation were then treated exactly the same as the products discharged from the drum in the pilot plant. That is, they were passed over the vibrating screen, sampled, and then subjected to the usual float-and-sink procedure used in evaluating plant performance. In effect, a float-and-sink bath was simply substituted for the dense-medium bath of the drum separator.

The data obtained in this way were then used to calculate the distribution figures shown in Table VIII. The effect of degradation was great enough to make the perfect separation of the float-and-sink bath appear poor. Even in the 2 to 1-in. size, 1.5 pct of the 1.50 to 1.60 sp gr fraction was found in the sink product, and 11.8 pct of the 1.60 to 1.70 sp gr

Table VII. Summary of Pilot Plant Performance

Coal	High-Density Tests						Low-Density Tests			
	Buffalo No. 5	Surmi	Raven	Wilkeson No. 3	Belling- ham	Nugget	Buffalo No. 2	Wilkeson No. 3	Nugget	Belling- ham
Test number	30	13	37	53*	41	35	31	54*	32	25
Specific gravity of separation	1.69	1.61	1.59	1.58	1.55	1.62	1.30	1.36	1.39	1.41
Pct ± 0.10 material in feed†	8.2	19.3	12.3	17.3	19.8	8.4	83.1	64.4	85.6	64.8
Ash in feed, pct	22.2	22.1	26.1	37.6	22.9	15.6	11.0	38.2	17.5	16.4
Ash in washed coal, pct	8.7	13.9	17.3	17.3	14.8	12.9	2.5	9.7	8.6	10.8
Ash in refuse, pct	66.3	49.6	61.5	68.4	55.4	56.2	22.3	48.4	37.5	26.1
Yield of washed coal, pct	76.5	77.1	80.3	60.2	80.0	93.5	57.3	26.2	69.3	63.6
Theoretical yield, pct**	76.6	78.2	80.3	60.3	80.1	94.4	61.0	29.3	71.2	65.5
Recovery efficiency, pct††	99.9	98.6	100.0	99.8	99.9	99.0	93.9	89.4	97.3	97.1
Sink in washed coal, pct‡	0.3	0.5	2.2	0.8	1.2	2.2	2.0	2.3	0.8	2.6
Float in refuse, pct§	0.6	0.9	1.4	1.2	2.4	0.9	8.5	3.2	4.5	7.3
Total misplaced material, pct§	0.9	1.4	3.6	2.0	3.6	3.1	10.5	5.5	5.3	9.9
Error area	19	19	38	21	15	102	13	18	14	13
Specific-gravity distribution, pct to washed coal										
Under 1.30	100.0	100.0	100.0	100.0	100.0	100.0	89.4	98.5	99.4	99.3
1.30 to 1.40	99.9	99.7	100.0	99.9	100.0	100.0	14.6	60.0	90.6	87.0
1.40 to 1.50	99.8	99.2	99.7	99.8	99.5	99.8	0.7	2.5	4.3	15.2
1.50 to 1.60	99.8	95.1	69.7	75.0	46.9	87.4	0.0	0.4	0.4	0.4
1.60 to 1.70	81.5	9.8	27.2	9.4	3.8	44.6	0.0	0.0	0.0	0.0
1.70 to 1.80	8.6	2.0	6.9	2.0	0.0	33.5	0.0	0.0	0.1	0.0
Over 1.80	0.4	0.6	1.2	0.2	0.0	18.2	0.0	0.0	0.0	0.0

* 2 to 1/8-in. feed.

† Pct of material within ± 0.10 of specific gravity of separation, adjusted for 2.0 material.

** Float-and-sink yield of feed at ash content of washed coal.

†† Yield of washed coal/theoretical yield $\times 100$.

§ At specific gravity of separation.

fraction appeared in the float product. With decreasing particle size the influence of degradation on the migration of material from one density fraction to another was even more pronounced. Considering all size together, the distribution data for the float-and-sink bath are actually inferior to the results obtained on this coal in the pilot plant when it was fresh. The error area for the float-and-sink separation was 24, while an area of only 15 was obtained on the fresh coal in the plant.

These results provide ample evidence to substantiate the contention that the poor performance of the pilot plant on Raven and Nugget coals was due to degradation. However, they have much broader significance. Many bituminous coals are more friable than the stored Bellingham material, and many of them contain at least as high a proportion of intergrown coal and impurity particles from which liberation is obtained on breakage. Thus comparable effects from degradation must be obtained in many washing tests. In fact, a similar trial showed that the degradation effect on Wilkeson No. 3, a friable coking coal containing only a normal amount of intergrown material, was an error area of 10. Although distribution data and error area are extremely useful criteria, it must be borne in mind that they are susceptible to abnormalities due to degradation. Small differences in error area are not

Table. VIII. Distribution Data for Test on Bellingham Coal in Float-and-Sink Bath

Sp Gr	Sizes Recovered in Float Coal, Pct				
	2 to 1	1 to ½	½ to ¼	¼ to 2	2 to 0
Under 1.30	100.0	100.0	100.0	100.0	100.0
1.30 to 1.40	100.0	100.0	100.0	99.8	99.9
1.40 to 1.50	100.0	99.8	99.7	99.2	99.7
1.50 to 1.60	98.5	96.6	95.5	92.6	95.8
1.60 to 1.70	11.8	16.2	16.7	18.7	16.6
1.70 to 1.80	0.0	2.8	2.6	7.5	5.6
Over 1.80	0.0	2.3	1.6	1.1	0.9
Error area	14	25	23		24

significant unless all factors having a bearing on sharpness of separation are strictly comparable.

It is noteworthy that recovery efficiency is less subject to errors owing to degradation than are either error area or percentage of misplaced material. The latter two criteria are affected directly by the liberation of particles accompanying degradation. On the other hand, liberation of impurity in a washed coal does not change its ash content, and therefore affects recovery efficiency only indirectly through the resulting minor change in the yield-ash curve of the composite feed to the washing unit.

Operating Variables

A number of factors influencing sharpness of separation effected between coal and impurity in the dense-medium bath are beyond the control of the operator, because they are fixed by either the design of the separating vessel or the nature of the cleaning problem. Other factors, however, are subject to some degree of control, and hence can be regulated to provide optimum conditions in the bath. Among the more important factors in the latter category that were investigated in the course of the present work are density control, density differential between the top and bottom of the bath, amount of agitation in the bath, congestion or crowding of the coal in the bath, and medium contamination.

Controlling the density of the medium proved to be the simplest of the problems encountered. During the first year of plant operation, density was measured in the conventional manner by periodical weighing of a flask of known volume. Then a device for continuously indicating density was developed

Table IX. Influence of Density Differential in Bath on Performance

Item	Test No.	
	15	16
Specific gravity of separation	1.58	1.57
Difference in specific gravity between top and bottom of bath	0.00	0.05
Distribution data, pct to washed coal		
Under 1.30	100.0	100.0
1.30 to 1.40	99.0	99.8
1.40 to 1.50	96.7	99.4
1.50 to 1.60	71.7	73.0
1.60 to 1.70	21.1	2.9
1.70 to 1.80	6.0	0.3
Over 1.80	1.0	0.4
Error area	37	17

to replace the time-consuming, cumbersome procedure of manual weighing. The densimeter,³ as it is known, is merely a 40-ft coil of ordinary garden hose suspended from one arm of a balance. Since the volume of medium in the hose is constant, any change in density produces a corresponding change in weight. Thus, if a tare weight on the balance is shifted to bring the indicating arm to 0 on the scale when the circulating medium is at the desired density, any subsequent change in density is indicated directly on the scale. The operator has merely to glance at it to know the direction and extent the density of the medium has varied from the desired value. With the use of this simple densimeter, the fluctuation in density during a test period has been reduced to thousandths rather than hundredths, and separation errors due to varying density have thus been eliminated.

Density Differential: Thickening of the medium, with a resultant differential in density between the top and bottom of the bath, may occur, depending on the amount of agitation provided by the rotation of the drum. Differential also is influenced by the density of the medium, the fineness of the magnetite, and the amount of clay present. While the density of the medium is fixed by the requirements of the separation, all other factors are subject to regulation. Thus density differential is one of the controllable variables. When operating at specific gravities of 1.55 and higher it appears to be of considerable importance, but at specific gravities of 1.40 and lower it has no influence on performance.

The density differential is determined by comparing the specific gravity of the medium overflowing the discharge weir with that of a sample of the medium collected just above the sink-lifting flights at the discharge end of the drum. The measurement is thus an arbitrary value referring solely to the conditions at the discharge end. It is entirely possible that the differential would be higher if it were measured farther toward the feed end, but no attempt has been made to measure it along the full length of the drum. Present measurement has proved a satisfactory criterion of bath conditions.

Table IX shows the results of two companion tests made on 2 to ¼-in. Surmi coal with and without density differential in the bath. Both tests were made under identical operating conditions except

for the cleanness of the medium. In test 15 the medium contained a great deal of clay originating from a coal run previously, and as a result there was no measurable differential. For test 16 the same medium was freed of clay by routing it through the medium-cleaning circuit to develop a differential of 0.05. Under this condition the performance was clearly superior. Rejection of impurity was improved greatly, and the recovery of coal was slightly better. The error area was decreased from 37 to 17.

Table X summarizes, in terms of error area, all the test data available for which operating conditions other than density differential were strictly comparable. These error areas are necessarily influenced to a limited extent by the nature of the impurities in the individual coals involved, because flaky or lower-density impurities are not so completely rejected as more amenable varieties under any bath condition. Nevertheless, in the high-density group of tests those of lowest error area were obtained with differentials of 0.04 or 0.05, whereas for the higher error areas no differential existed. It is equally significant that in the low-density tests differential is unimportant.

Presumably density differential allows the bed of float coal at the surface of the bath to dilate vertically. This dilation or distention minimizes mechanical entrapment of sink particles, and thus improves the sharpness of the separation. When operating at low densities, the relative buoyancy of the coal is less. Therefore there is less tendency to form a compact bed favoring mechanical entrapment; hence density differential is less important. An analogous condition can be observed in float-and-sink testing. When a sample is immersed in a high-density bath the buoyancy of the coal is so great that a firm mat or layer of float forms at the surface immediately; vigorous stirring is required to

density differential in the bath. An undesirably high differential occasioned by over-cleaning the medium can be reduced by increasing the drum speed; while with a more stable medium, differential can be increased by slowing down the drum.

Obviously, violent agitation of the bath would militate against sharpness of separation because of the uncontrolled, stray currents produced. Conversely, however, too quiescent a bath promotes mechanical

Table XI. Relation Between Drum Speed and Sharpness of Separation

	Test No.		
	13	16	11
Drum speed, rpm	1	2	5
Specific gravity of separation	1.61	1.57	1.55
Distribution data, pct to washed coal			
Under 1.30	100.0	100.0	100.0
1.30 to 1.40	99.7	99.8	99.7
1.40 to 1.50	99.2	99.4	95.1
1.50 to 1.60	95.1	73.0	45.9
1.60 to 1.70	9.8	3.3	9.2
1.70 to 1.80	2.0	0.5	2.0
Over 1.80	0.6	0.3	0.4
Error area	19	17	22

entrapment; returning to the analogy of the float-and-sink test, some stirring is essential. Between these two extremes there is considerable latitude in the amount of agitation that can be tolerated without impairing the separation.

Table XI shows the distribution data and error areas for three tests made on the same coal at drum speeds of 1, 2, and 5 rpm. Size of feed, feed rate, and density differential were the same in all three tests, and the range in specific gravity of separation from 1.55 to 1.61 is not great enough to preclude direct comparison. From the standpoint of both distribution data and error areas, the test at 2 rpm is slightly superior to the others, but the differences are not great.

At 5 rpm the surface of the bath in the drum is agitated so vigorously that small transverse waves are formed. It is somewhat surprising that such a degree of agitation does not impair the sharpness of separation. However, four tests have been made at 5 rpm and six have been made at 3 rpm, and except where the resulting density differential was insufficient, the results of these runs were not inferior to those obtained at the normal operating speed of 2 rpm. Thus the conclusion that considerable agitation can be tolerated appears to be well founded.

Owing to the higher peripheral speeds involved, a commercial-size drum might have to operate at a much lower rpm than the small laboratory unit to develop the same degree of agitation. Nevertheless, the same relation between agitation and sharpness of separation should apply.

Ratio of Medium to Coal: The bath of medium transports the float material forward through the drum and discharges it over the weir. Thus the depth of medium overflowing the weir must be great enough to remove coal at the rate required to preserve a degree of bed congestion conducive to good gravity separation. Primarily, the depth at the weir is determined by the feed rate, the percentage of float, and the size of the coal. Higher feed rate, a greater proportion of float, or coarser coal requires greater depth. Excessive depth over the weir is to be avoided, because it increases the horizontal velocity of the medium, reducing retention time.

Table X. Comparison of Tests with High and Low Density Differentials

Test No.	Separation, Sp Gr	Sp Gr* Differential	Error Area
High-density tests			
41	1.55	0.05	15
40	1.57	0.04	16
16	1.57	0.05	17
53	1.58	0.05	21
14	1.60	0.00	28
39	1.50	0.00	35
15	1.58	0.00	37
37	1.59	0.00	38
Low-density tests			
34	1.35	0.00	11
25	1.41	0.03	13
27	1.39	0.03	14
29	1.39	0.00	14
32	1.39	0.00	14
33	1.39	0.00	14
50	1.39	0.05	15
51	1.39	0.04	17

* Measured between top of bath and just above sink-lifting flights at discharge of drum.

release entrapped particles of sink material. With a low-density bath, however, the particles are less buoyant, and the layer of float is distended by only gentle agitation.

Drum Speed: The primary function of drum rotation is to elevate sink material, but an important secondary effect is the agitation of the bath. Thus the speed of drum rotation can be used to control agitation, which in turn assists in controlling the

Table XII shows clearly the results of operating the drum with an inadequate depth overflowing the weir. Tests 17 and 18 were made on identical feeds at 5.2 tons per hr, nearly three times the normal feed rate. In test 17 the usual weir depth of 1½-in. was preserved. This caused excessive crowding at the weir, and resulted in clean coal being forced down

Table XII. Influence of Congestion on Sharpness of Separation

Item	Test No.	
	17	18
Feed per hr, tons	5.2	5.2
Depth of medium overflowing weir, in.	1.13	1.25
Medium overflowing weir, gpm	23	26
Feed per hr per sq ft of bath, tons	2.50	2.50
Feed per gal of medium, lb	7.54	6.67
Distribution data, pct to washed coal		
Under 1.30	93.3	100.0
1.30 to 1.40	89.5	99.7
1.40 to 1.50	82.7	97.9
1.50 to 1.60	53.7	68.7
1.60 to 1.70	5.9	3.6
1.70 to 1.80	1.3	0.0
Over 1.80	0.0	0.0
Specific gravity of separation	1.56	1.57
Error area	39	20

into the sink-lifting flights where it was lost. Increasing the flow of medium over the weir from 23 to 26 gpm in test 18 ended bed congestion, eliminated loss of clean coal, and reduced the error area of the separation from 39 to 20.

It is noteworthy that the cleaning results in test 18 are as good as those obtained on the same coal at the normal feed rate of 1¾ tons per hr. The throughput at which performance of the plant would suffer has not been reached.

The ratio of coal to medium overflowing the weir is a measure of the degree of bed congestion, and therefore reflects the likelihood for sink particles becoming mechanically entrapped in the float product. In test 17, with 7.54 lb of coal per gal of medium, there was poorer elimination of 1.60 to 1.80

Table XIII. Comparison of Tests of 2 to ¼-in. Bellingham Coal with Clean and Contaminated Media

Item	Test No.	
	43	44
Specific gravity of medium	1.60	1.60
Nonmagnetic material in medium solids, pct	47.2	9.9
Specific-gravity differential in bath	0.00	0.01
Distribution data, pct to washed coal		
Under 1.30	100.0	100.0
1.30 to 1.40	99.9	99.9
1.40 to 1.50	99.9	99.8
1.50 to 1.60	93.4	86.6
1.60 to 1.70	29.7	23.8
1.70 to 1.80	6.0	5.6
Over 1.80	1.9	2.3
Specific gravity of separation	1.62	1.61
Error area	29	31
Recovery efficiency, pct	99.5	99.5

sp-gr material than in test 18 at 6.67 lb. Thus optimum congestion for maximum throughput without impaired performance appears to be 6½ to 7 lb of coal per gal for a 2 to ¼-in. feed.

Contamination of Medium: The clay associated with the Bellingham coal disintegrates readily in water, and hence is a marked source of medium contamination. When the pilot plant was run on this coal, provision was made to divert sufficient medium to the medium-cleaning circuit to insure the maintenance of satisfactory quality. Cleaning only the medium from the rinsing section of the

screen did not maintain the desired density differential in the bath. Moreover, it was feared that an excessive amount of clay in the medium might increase its viscosity enough to interfere with the sharpness of separation.

Once satisfactory cleaning results had been obtained with clean medium, however, a run was made with dirty medium to see how much the performance of the plant would suffer. In preparation for this run, several tons of refuse from a previous test was recirculated through the plant until most of the clay had disappeared. The medium obtained with this intentional contamination was so stable that it would not settle out on standing for hours, and magnetic separation in a Davis tube showed that 47.2 pct of the solids were nonmagnetic. Preliminary experiments had shown that when 55 pct of the medium solids were Bellingham clay a semi-gel that would scarcely pour was formed. Thus it was felt that operation of the plant with the contaminated medium would show, in exaggerated degree, the influence of viscosity.

Test 43 was made with the contaminated medium and test 44 with clean medium to provide a direct comparison. As shown in Table XIII, the distribution data and error areas for the two tests are very similar. The recovery efficiencies are identical. These tests were made on the Bellingham coal after it had been weakened by storage, and because of the degradation factor discussed previously, neither one appears to be as sharp a separation as that obtained on the fresh coal. Despite this mitigating circumstance, the performance obtained with dirty medium clearly is not significantly inferior to that with clean medium under otherwise similar operating conditions.

One test, even with badly contaminated medium, is not conclusive, and a number of writers have expressed the opinion that the viscosity of medium is a factor in performance. It would be presumptive, therefore, to conclude that viscosity is unimportant. In view of the present results, however, it would be interesting to see comparable figures for a commercial plant. Perhaps viscosity has been overrated.

At least in the pilot plant, medium contamination is more important from the standpoint of stability than from its influence on viscosity. Too stable a medium will not provide the density differential generally required in the separating bath for best cleaning results.

Summary

Seven coals exhibiting a wide range in washability characteristics have been washed in a small-scale, dense-medium pilot plant. Separations ranged from 1.30 to 1.69 sp gr. With most of the coals treated at the higher densities, virtually perfect recovery of material lighter than 1.40 sp gr was obtained. Rejection of impurity was not perfect, however, with as much as 0.6 pct of the sink 1.80 material entering the washed coal. Operation at specific gravities of 1.41 and lower was characterized by perfect elimination of impurity, but with impaired recovery of the lightest coal.

Recovery efficiencies varied from 98.6 to 99.9 pct for separations made in the usual range of washing densities. Decreased efficiencies were obtained in separations at lower densities. With one particularly dirty coal, which provided a yield of clean product amounting to only 26 pct, the efficiency was only 89.4 pct, despite the fact that the separation between coal and impurity was relatively sharp. Results

similar to this might be expected in re-treating refuse by dense medium, because of the low separating density required and the low yield of clean product.

With operation at all densities, the smaller sizes of the feed were not separated as sharply or efficiently as the coarser material. Similarly, impurities of the tabular shape were not removed as completely as others, and impurities of relatively low density such as carbonaceous shale and bone were not removed as completely as heavier varieties, such as true shale.

A differential in specific gravity of about 0.05 between the top and bottom of the separating bath gave the best cleaning results when operating at the higher densities, but was unimportant in low-density separation.

A great deal of latitude was found in the degree of agitation of the bath that could be tolerated without impairing performance. Drum speeds of 1 and 5 rpm gave cleaning results nearly as good as those obtained at 2 rpm, indicating the need for some agitation of the bath.

Comparative washing tests were made with clean and badly contaminated mediums to determine whether the viscosity of a medium influences cleaning results. Even though the contaminated medium

contained 47.2 pct nonmagnetic material, largely clay, it gave washing results that were not significantly inferior to those obtained with clean medium.

Anomalous results obtained in cleaning several coals were traced to degradation of the washed products, which caused a marked increase in error area but had little influence on recovery efficiency.

Acknowledgments

The work described in this report is part of a continuous program of coal-preparation research sponsored jointly by the Northwest Experiment Station, Region II, Bureau of Mines, and the School of Mineral Engineering, University of Washington. The program is conducted under the general direction of L. C. McCabe, Chief, Fuels and Explosives Division, Bureau of Mines.

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An Analysis of Mine Opening Failure By Means of Models

by Bernard York and John J. Reed

Mine opening stability was investigated by loading model openings to failure. Eight-inch plaster blocks were cast with small uniform section openings passing through the centers. After curing, the models were jacketed and loaded hydrostatically in a closed vessel. Opening size, shape, and orientation were varied to determine how these factors influence opening stability.

THE investigation described in this paper is a continuation of a project begun in 1946 by Bernard York and W. E. Bush. The experimental method used in the investigation involves the hydrostatic loading of a model mine opening to failure. Most investigators agree that this loading closely approaches actual conditions in very deep mines. The conditions are further idealized by using an essentially isotropic model material. The experiments consist of casting 8-in. plaster cubes, or 8-in. diam cylinders 8-in. long, having a relatively small uniform section hole passing through their centers. After thorough drying, the models are jacketed in a flexible plastic bag and loaded hydrostatically with

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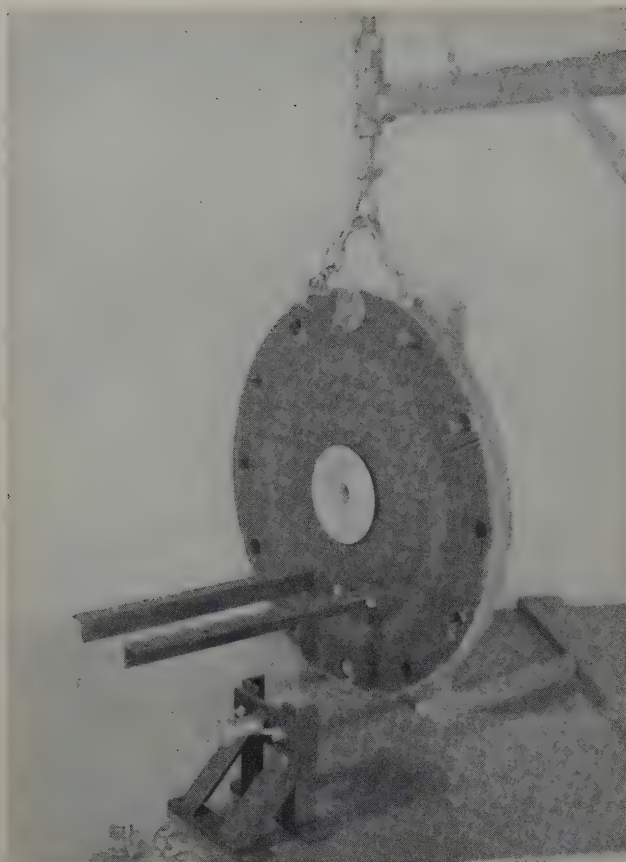
Discussion on this paper, TP 3559A, may be sent (2 copies) to AIME before Sept. 30, 1953. Manuscript, Oct 17, 1952. Los Angeles Meeting, February 1953.

The work described in this paper was done in partial fulfillment of the requirements for the degree of Master of Science in Mining at the University of California.



Fig. 1—Pressure vessel open in position to jacket the model.

water in a closed vessel. At about 300 to 500 lb per sq in. pressure the central opening fails. The fracture pattern resulting from opening failure is studied in a plane normal to the longitudinal axis of the opening. The model block is sawed through its center after loading to expose this plane and to eliminate the possibility of end effect. By changing



the size, shape, and orientation of the opening it is possible to determine how the overall stability of an opening varies as a function of these factors. Once the actual manner of failure of mine openings is thoroughly understood, the means to prevent such failure may become evident. The experiments described in this paper, which are performed under the most simplified conditions, are a beginning toward this end.

In analyzing mine opening failure it is important to consider *how* an opening fails, *where* initial failure occurs, and *when* failure occurs as the load is increased. This last factor is the true measure of mine opening stability.

The results of this investigation substantiate mathematical and photoelastic analyses in demon-

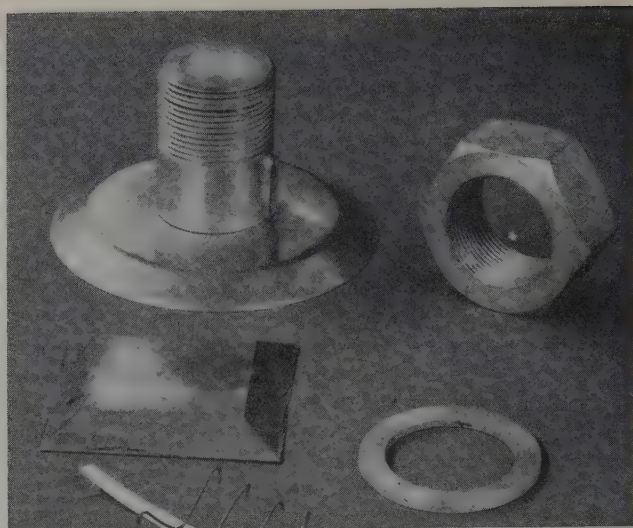


Fig. 2a (left)—Observation tube assembly in place, and Fig. 2b (right) observation tube assembly and rear cover plate.

strating *how* mine openings will fail under hydrostatic loading. They will fail in shear. Experimental results also agree with theoretical analysis as to *where* initial failure owing to stress concentration will occur on non-cylindrical openings. However, there is a distinct difference between test results and theoretical analysis as to *when* various sizes and shapes of openings will fail in a brittle material loaded hydrostatically.

As described in the section on the factors controlling opening stability, this difference can be explained in the test results on the models by consideration of probability of failure as well as the calculated maximum stresses on the openings.

If the probability of failure has such an important control on the stability of a nearly perfect cast opening in a relatively isotropic model material, then it certainly must be the major controlling factor on the stability of a prototype mine opening; because the rock is never isotropic, the opening is usually highly irregular, and, because of blasting, it is surrounded invariably by a zone of shattered material with a high failure-probability.

Using present methods and equipment, the stability of mine openings can be increased considerably by careful design of the drill rounds and blasting techniques used to create them. More trim holes of smaller diameter, loaded lightly with a low-grade bulky explosive, will result in the minimum damage to the remaining rock section.

The ideal method of driving an opening would be by boring or cutting out the rock without blasting. Shafts drilled by shot drills are an example, and have proved practical. Unfortunately, shot drills will drill only near-vertical openings, and a machine to drill or cut a horizontal opening of usable size in hard rock remains to be developed, although horizontal holes 21 in. in diam have been drilled by diamond drill. An opening driven by such means would not need to be as large as conventional mine openings because of its smooth self-supporting nature. A round opening 4 or 5 ft in diam might be quite effective. Scrapers and conveyors are capable of moving large tonnages through openings of small cross section. The development of a machine to bore or cut horizontal mine openings is worthy of the full support and interest of the mining industry.

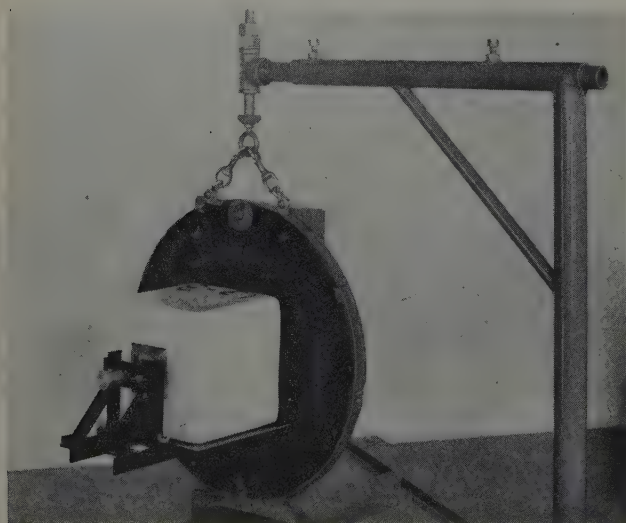


Fig. 3—Rack and unjacketed model mounted on head of pressure vessel.

The importance of making mine openings small, with regular cross section, without damaging the remaining rock section, has been neglected by mine operators generally. The results of this investigation demonstrate that these factors exert the major control on the stability of the resultant opening. Artificial opening support is expensive and troublesome. Therefore it is important to design the opening carefully as to size and shape for maximum stability, and then exert every effort to drive the opening in such manner as to preserve the inherent strength of the material as fully as possible.

Method of Investigation

Model Material: A model material was required for the investigation which would fail brittly as do most rocks. A plaster mixture was found suitable, and a standard mix of 1.0 part No. 1 pottery plaster, 7.0 parts —200 mesh silica flour, and 3.1 parts water has been adopted. The average uniaxial compressive strength of this mixture is about 300 psi. By standardization and accurate control of mixing, pouring, and curing procedures very satisfactory uniformity of strength and properties is obtained from block to block.

Loading the Models: A general view of the pressure-loading equipment is shown in Fig. 1. The pressure vessel is a section of 12-in. pipe with a bolted steel cover. A rack fastened to the inside of the vessel head carries the model block, Fig. 2a. A round observation tube passes through the head of the vessel and provides a 1-in. round opening directly opposite the opening in the model, giving an unobstructed view directly inside the model opening during loading. The observation tube assembly with its bearing plate is shown detached in Fig. 2b, and in place in the vessel head, Fig. 2a. The square beveled plate shown in Fig. 2b covers the opposite end of the model opening. A spring-loaded plunger, shown detached in Fig. 2a, bears on the rear cover plate and holds the block in close contact with the plate of the observation tube until the pressure comes on. Fig. 3 shows an unjacketed model mount-

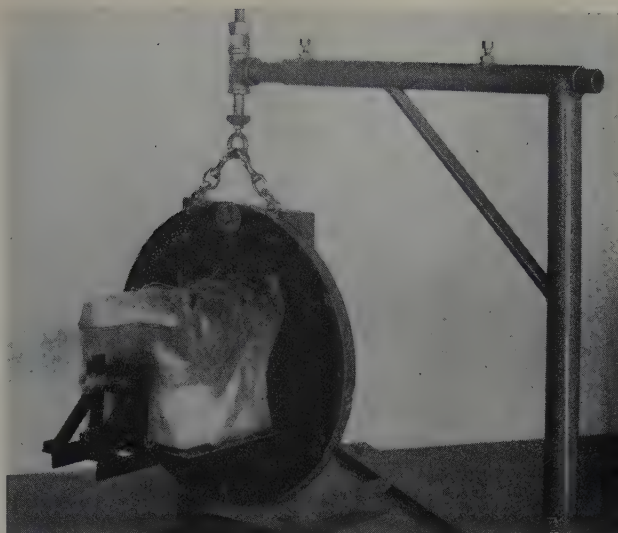


Fig. 4—Jacketed model ready to enter pressure vessel.

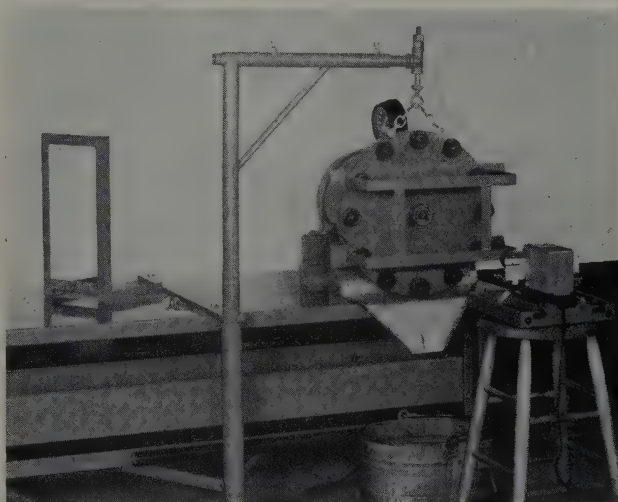


Fig. 5—Vessel closed with protective shield and light source in position.

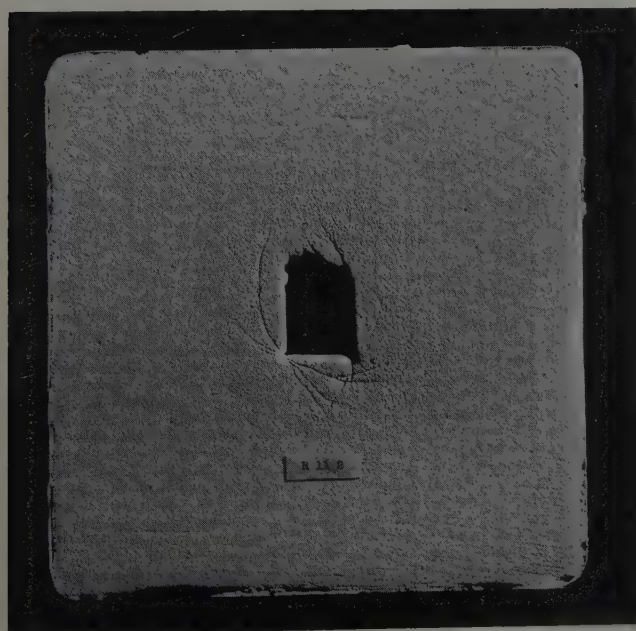


Fig. 6a—A photograph of model type R-11, which has an opening 1 in. square. Fig. 6b at left shows a close-up of the localized fracture pattern. All models are made of pottery plaster, —200 mesh silica flour, and water.



Fig. 6b—A close-up of fracture failure, model type R-11. Larger and smaller openings of similar shape failed in exactly the same manner, except that loading pressure at initial failure was lower and higher, respectively.

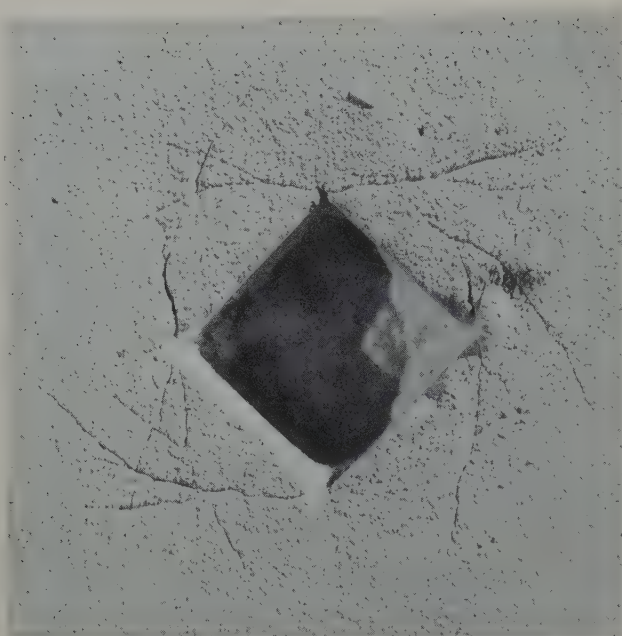


Fig. 7—Model type R-7 with opening 1 in. square. Model type R-19 has opening of similar shape, 0.35 in. square. Both models are square.



Fig. 8—A photograph of model type R-10, showing opening 1.4 in. round. The model is square. Similar models have openings 0.5 in., 0.75 in., 1 in., and 2 in. round.

ed on the rack and Fig. 4 a jacketed model ready to enter the pressure vessel. The jacket is a simple flat bag 20 by 30 in. made commercially from 0.008-in. vinyl plastic material. The bag is open at one end, and once the model is inside, the mouth of the bag is sealed for about 1 in. with vinyl-to-vinyl cement. The observation tube passes through a hole in the side of the bag, and the bag itself forms a gasket between the bearing plate and the head of the vessel. Fig. 5 shows the equipment with the vessel closed and ready to apply pressure. A rigid transparent shield protects the observer from plaster and water which might blow out through the observation tube if the jacket fails under pressure. A slide projector provides illumination within the model opening during loading.

The pump connected to the pressure vessel is operated manually, and is capable of developing pressures in excess of 2000 psi. During loading of the model, one person operates the pump and watches the pressure gage, while a second person observes the model through the observation tube. The pressure is raised very slowly at first to allow all possible air to work out of the collapsing plastic bag and escape through the observation tube. Then it is gradually increased with a short pause after each 25-lb increment. The pressure at initial failure is recorded when the first definite spalling is observed within the opening. This pressure can be determined within the nearest 25 psi, or about 7 pct of the ultimate load. Thus the limit of accuracy of the method as described is about 7 pct.

Types of Models and Openings Tested

Examples of some of the 11 types of model blocks tested during this investigation are shown in Figs. 6-10, which illustrate the fracture failures formed during testing. Two or more blocks were made of each type. A complete series of square and round openings was tested. Dimensions of the openings ranged from 0.5 to 2.0 in.

It became apparent early in the investigation that the maximum dimension of the opening was an

important factor controlling the loading pressure required to cause failure. Therefore corresponding round and square openings were made, the diameter of the round openings being equal to the diagonal of the square ones. Thus the relative stability of the two shapes could be compared.

The results of testing 44 model blocks indicate that they are sufficiently uniform in strength and physical properties to compare the manner and degree of failure, and the loading pressure at failure, from block to block. Throughout the tests, duplicate blocks invariably gave duplicate results.

Only the tests of 1.4-in. openings will be de-



Fig. 9—Model type R-15, which has opening 1 in. square. The model itself is circular.

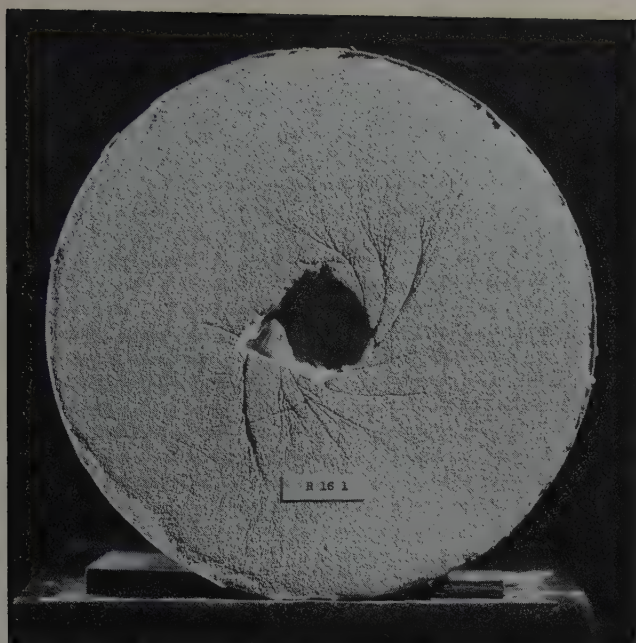


Fig. 10a—A photograph of model type R-16, which has an opening 1.4 in. round.



Fig. 10b—A close-up of the fracture failure around opening of model type R-16, shown at left.

scribed in detail. Larger and smaller openings failed in exactly the same manner except that the loading pressure at initial failure was lower and higher, respectively. The photographs of fracture failures, Figs. 6-10, were all taken on sections through the center of the blocks and normal to the axes of the openings. For two typical models, Figs. 6a and 6b and Figs. 10a and 10b, both a general picture of the entire block and a close-up of the localized fracture pattern are included.

Mathematical analysis of the stresses around an opening in a hydrostatic stress field shows that the stress tangent to the periphery of the opening is everywhere a compressive stress. This is in accordance with the theory of elasticity developed by Timoshenko. It can be shown by the theory that if the unit hydrostatic load is P , the maximum unit compressive stress at the periphery of a cylindrical opening will be $2P$. The maximum unit shear stress will be P acting on a curved plane starting at approximately 45° with the periphery of the opening. Under hydrostatic loading these stresses are the minimum on a cylindrical opening and are equal anywhere on the periphery of that opening. Around non-cylindrical openings the stresses will be lower than $2P$ in some portions and higher in others and will be highest at points where the radius of curvature is smallest.

This same problem of stress analysis can be treated experimentally by photoelastic methods, and the same result is obtained as by mathematical analysis.

The manner in which the model openings failed supports both the mathematical and experimental stress analyses with regard to *how* and *where* the openings fail. Failure occurred along planes of maximum shear stress in every test, and the fractures exhibit the spiral pattern of the curves plotting that stress. There were no signs of tension failure in any models. Fig. 11 shows fracture fragments formed by failure of 1-in. square openings. They have the characteristic shape indicating failure in shear, and because of longitudinal translation of the

fracture, they are usually unbroken through the entire length of the opening.

Both the cylindrical and cubic models show the same fracture patterns on the various openings, indicating that in either model shape the opening is subject to approximately the same stress field. Initial failure in square openings occurred at corners because of stress concentration at these points.

Factors Controlling Opening Stability

Type of Loading: With existing equipment, only hydrostatic loading can be developed. With equipment capable of loading unequally on the two axes perpendicular to the axis of the central opening, it will be possible to influence opening stability by the type of loading employed.

Orientation of Opening: Under hydrostatic loading any unit plane within the model is subject to the same pressure, regardless of its orientation. Therefore any opening will be equally stable in a given hydrostatic stress field, regardless of its orientation.

Shape of Opening: According to the theory of elasticity, the shape of an opening in a hydrostatic stress field is the all-important factor that determines the maximum stress on the opening, and from



Fig. 11—Fracture fragments from 1-in. square openings.

this it is inferred generally that the shape of the opening is also the major factor controlling the stability of the opening.

The stress concentration at each corner of a true rectangle, one with sharp corners, is infinite. Even on a square opening with rounded corners, the maximum stress at the corners exceeds twice that on a circular opening.

The square openings used in this investigation all had relatively sharp corners. Therefore, according to the theory and inference cited above, there should be a wide difference between the loading at failure for a square opening and that for a round one. It appears highly significant that no such difference is apparent in the experimental results. Within the accuracy of the method, this investigation shows no appreciable difference in the loading at failure between a square opening and a round one of the same maximum dimension.

Size of Opening: The experimental results of this investigation indicate that the maximum dimension of the opening is the major factor controlling opening stability. Within the accuracy of this method, all openings of the same maximum dimension failed at the same loading pressure regardless of their shape. Fig. 12 is a graph of the test results made by plotting the greatest dimension of the opening against the loading pressure at initial failure. Results show that the loading pressure at initial opening failure will vary inversely with the greatest dimension of the opening. This result, coupled with field observations in mines, offers strong support to the argument that opening size is the major factor in controlling opening stability.

Mention should be made of the indicated compressive stress at failure for each of the various size openings tested. According to the elastic theory, this unit stress on a cylindrical opening is twice the unit loading pressure. Therefore the indicated unit compressive stress at failure on the periphery of the round openings tested varied from 634 psi on a 2-in. opening to 1034 psi on a 0.5-in. opening. This apparently anomalous result can be explained by several factors, as follows.

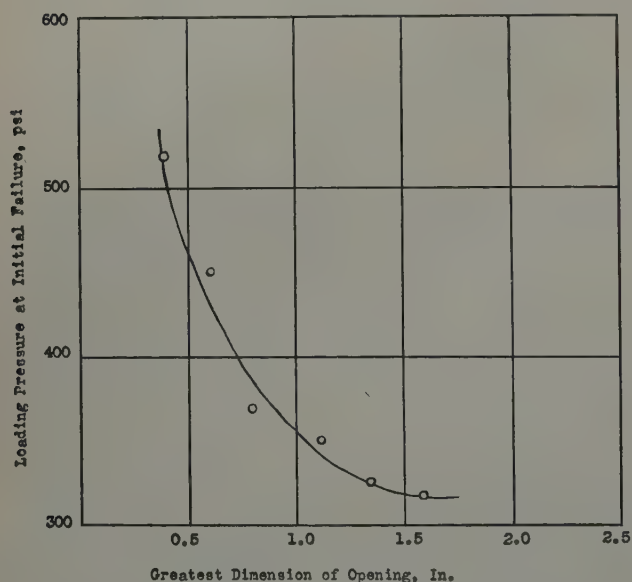


Fig. 12—A graph showing loading pressure at initial opening failure.

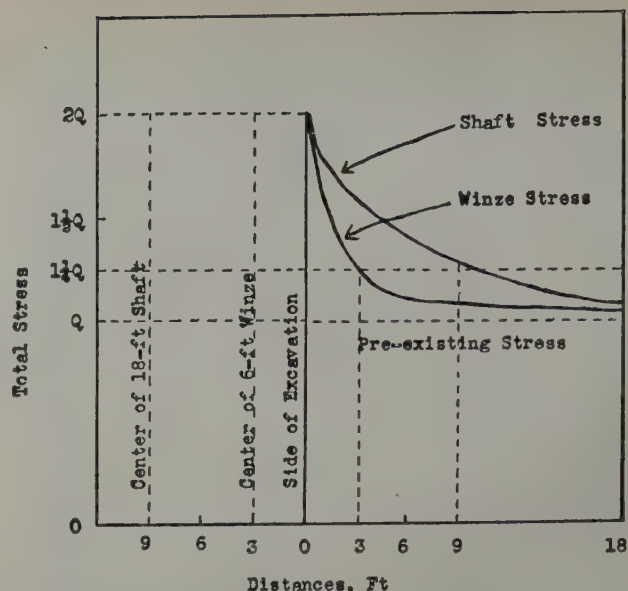


Fig. 13—Control of stress distribution by size of opening. (After Jack Spalding, *Deep Mining*, p. 91. Mining Publications, Ltd.)

1—Failure of a brittle material occurs in response to stress over a *finite length* of an incipient crack or weakness. The maximum stress calculated by the elastic theory is the stress at a *point* on the periphery of the opening. The *average* stress over a *finite length* of a crack must be considered. The larger the opening, the more gradual the stress gradient back into the walls, and therefore the higher the average stress on a given finite length of crack. Therefore, although the maximum stress at a point on the wall may be the same in both a large and a small opening, the larger opening will be less stable, because the given length of crack will be subject to a higher average stress. Fig. 13 shows this condition graphically for a winze 6 ft in diam compared with a shaft 18 ft in diam.

2—The probability for the occurrence of incipient cracks or other weaknesses in the material must be considered. This may be a factor of the volume of material involved, and also the size of the cracks, but a simple comparison of the opening perimeters should give a convenient measure of sufficient accuracy. Thus a larger opening will have a larger perimeter, with more probability of containing weaknesses, and will fail at a lower loading pressure. The difference between the perimeters of a square and a circle of equal greatest dimension is apparently less than the accuracy of the tests. Therefore the graph of Fig. 12 was drawn on the basis of greatest opening dimension alone.

3—The lack of appreciable difference in failure pressures for a square opening and for a round one of the same maximum dimension must be explained. On about two-thirds of the periphery of a square opening in a hydrostatic stress field, the stress is less than that on a round opening. Hence, on that portion of the square opening the probability of failure is less than on a round opening because the stress is lower. Although the stress at the corners of the square may be very high, the probability of a crack of important size occurring *at the corners* is very low, and the lower overall probability of failure raises the stability of the square opening into the same range as that of a circular one.

The Production of Sodium Sulphate From Natural Brines at Monahans, Texas

by William I. Weisman and Ross C. Anderson

THE manufacture of anhydrous sodium sulphate or salt cake from natural deposits in the United States has been in general somewhat of a marginal undertaking. Competition from foreign sources and from large quantities of byproduct sodium sulphate produced domestically in the manufacture of hydrochloric acid and other chemicals has existed and continues.

For example, most of the sodium sulphate produced is a byproduct or co-product in the manufacture of hydrochloric acid through the reaction of sodium chloride with sulphuric acid. In recent years, many manufacturers of rayon have installed equipment to recover sodium sulphate from waste spin bath liquors; today this is an important source.

Before World War II large quantities of sodium sulphate were imported from Germany. In 1949 imported material from Europe again appeared on the domestic market. Natural sodium sulphate from Canada in substantial quantities also enters the United States markets.

Despite this kind of competition, numerous attempts have been made to exploit various natural deposits of sodium sulphate in this country, but only a very few of these have survived economically over a period of years. One of these few operations is the plant of the Ozark-Mahoning Co. located 13 miles south of Monahans in West Texas. Several factors contributing to the successful life of this plant may be summarized as follows:

1—Geographical location. Monahans is reasonably close, freightwise, to the Kraft paper mills in Texas, Arkansas, and Louisiana; the Kraft paper industry is the greatest consumer of sodium sulphate in the United States.

2—Availability of natural gas as low cost fuel. Proximity of the natural gas fields of West Texas has been a tremendous asset, as the availability of low-cost natural gas is to all industry throughout the Southwest.

3—The nature of the deposit. The occurrence of sodium sulphate brines in southeastern New Mexico and West Texas has been very well described by Lang,¹ who writes that the brines are found in the Castile formation of the Delaware basin. Here weathering has altered the anhydrite so that a relatively porous gypsiferous zone overlies a dense impervious mass of anhydrite. This porous zone provides traps where percolating ground waters that have picked up soluble salts may lodge. These traps or pockets are the natural brine reservoirs exploited at Monahans. Although several hundred wells have

been drilled, currently some 25 wells serve to supply brine to the plant. All are within 1½ miles of the plant and are conveniently tied together by an electric power system serving electric motors driving the pumps.

Having the raw material in the form of a brine which can be pumped from shallow wells makes possible much simpler and more efficient handling than if it were in form of solids.

By contrast, other deposits of sodium sulphate, such as those in Arizona, Nevada, and North Dakota, are in the form of the solid minerals, thenardite and mirabilite, which present somewhat more of a mining and mineral dressing problem.² The largest producer of sodium sulphate from natural sources in the United States is at Searles Lake, Cal., and there a brine also is utilized.

4—Water. Substantial quantities are needed for cooling towers and for operation of gas engines. An area underlain with brine is not a promising source of fresh water, but fortunately, after a long search, an adequate supply was found nearly two miles from the plant.

It may be appropriate to discuss briefly the grades of sodium sulphate offered on the market. Salt cake is the name usually applied to the grade of sodium sulphate used by the Kraft paper industry. It may be a low analysis byproduct, 95 to 97 pct sodium sulphate, with as much as 1½ to 2 pct residual acid, or it may be a natural product. Usually salt cake is considered a low grade product, but a great deal of a higher grade of material is marketed under this name. The specifications for glassmakers' salt cake are somewhat higher than those of the paper industry, usually requiring 98 pct sodium sulphate.

Technical anhydrous sodium sulphate is a high grade material and usually exceeds 99 pct sodium sulphate. It finds the biggest market in the textile industry and is used as a builder in some synthetic detergents.

Glauber's salt, $\text{Na}_2\text{SO}_4 \cdot 10\text{H}_2\text{O}$, is usually of high purity. Preferred for some uses, it normally has been recrystallized from an anhydrous salt.

A unique manufacturing process has been developed at Monahans. This process results in the production of an exceptionally high grade of salt cake, and qualifies for nearly all uses, including many which specify the technical anhydrous grade. All of the finished product, which is very white, passes a 10-mesh U. S. Standard screen, and is retained on a 200-mesh U. S. Standard screen. It is over 99 pct Na_2SO_4 with main impurities being sodium chloride and magnesium sulphate. Iron content is less than 0.01 pct.

As mentioned, the raw material at Monahans is a brine drawn from wells. Attention was first attracted to this location because a so-called alkali

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Discussion on this paper, TP 3543H, may be sent (2 copies) to AIME before Sept. 30, 1953. Manuscript, Nov. 11, 1951. Revised Oct. 29, 1952. Norman, Okla. Meeting, September 1951.

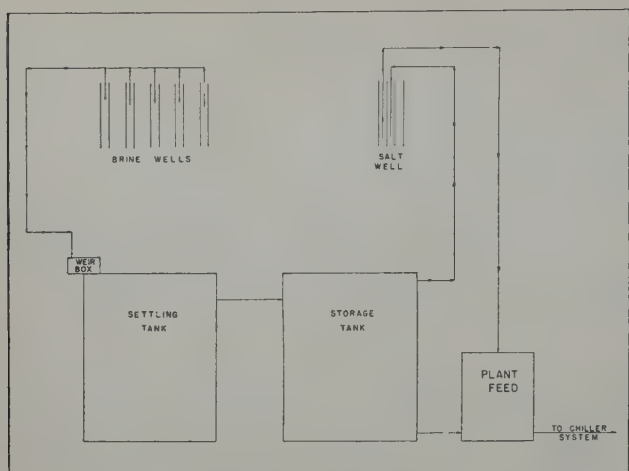


Fig. 1—Flowsheet illustrating handling of brine.

lake here was noted to be rich in sodium sulphate. In 1931 and 1932 investigation and drilling around the shore of the lake and in the general area revealed the existence of the brines. These offered a raw material of uniform nature in comparison to the lake brine, which varies greatly in concentration depending on seasonal factors of rain or drought.

Nevertheless, the lake may be utilized to a certain extent in processing the brine. Actually, two different brines are drawn from two different levels in the wells. At a depth of 60 ft there occurs an abundant supply of weak brine, 1.160 sp gr, and 7 to 8 pct sodium sulphate. In summer brine from this level is pumped onto the lake, where it is concentrated by solar evaporation to 1.180 sp gr, or about 10 pct sodium sulphate. Brine at this concentration is excellent as feed to the plant. Pumping of brine is controlled to maintain a constant level in the lake.

At a depth of 90 ft a strong brine of 1.180 to 1.20 sp gr is found. This brine is used directly as plant feed in winter and supplements the lake in summer. The deeper level has never produced so abundantly as the shallower levels; hence every effort is made to conserve the stronger brine. This brine averages 11.2 pct sodium sulphate with a typical analysis showing 6 pct chloride, 8.2 pct sulphate, 1.8 pct magnesium, and sufficient sodium to satisfy stoichiometric requirements. Traces of other elements are present.

The strong brine wells are pumped at rates of 3 to 18 gal per min. More rapid pumping will cause the brine strength to decrease. The optimum rate of pumping for a given well is determined from experience. Occasionally one of these wells is abandoned because the brine becomes dilute. It is uneconomical to continue to utilize weak brine from the 90-ft level when there is ample weak brine at shallower level.

Pumps are F. E. Meyers pump jacks with a 12-in. stroke, driven by 840-rpm, 2-hp, 220-v motors. The process used in the plant consists of two principal steps. The first is to cool the brine to crystallize out Glauber's salt, which is 56 pct water and 44 pct sodium sulphate. The second step is to melt the Glauber's salt, evaporate the water of hydration, and dry to anhydrous sodium sulphate.

It would be impossible to dehydrate the brine directly because, aside from the considerable fuel cost for evaporation, magnesium sulphate and sodium chloride are present in substantial quantities and a mixed salt would be obtained. Glauber's salt has a very steep solubility curve, and excellent recovery is made by the crystallization method used.

As a matter of fact, sodium chloride is added intentionally to the raw brine before chilling; the result is a further decrease in the solubility of the Glauber's salt. This is accomplished by pumping about one-third of the raw brine into a deposit of

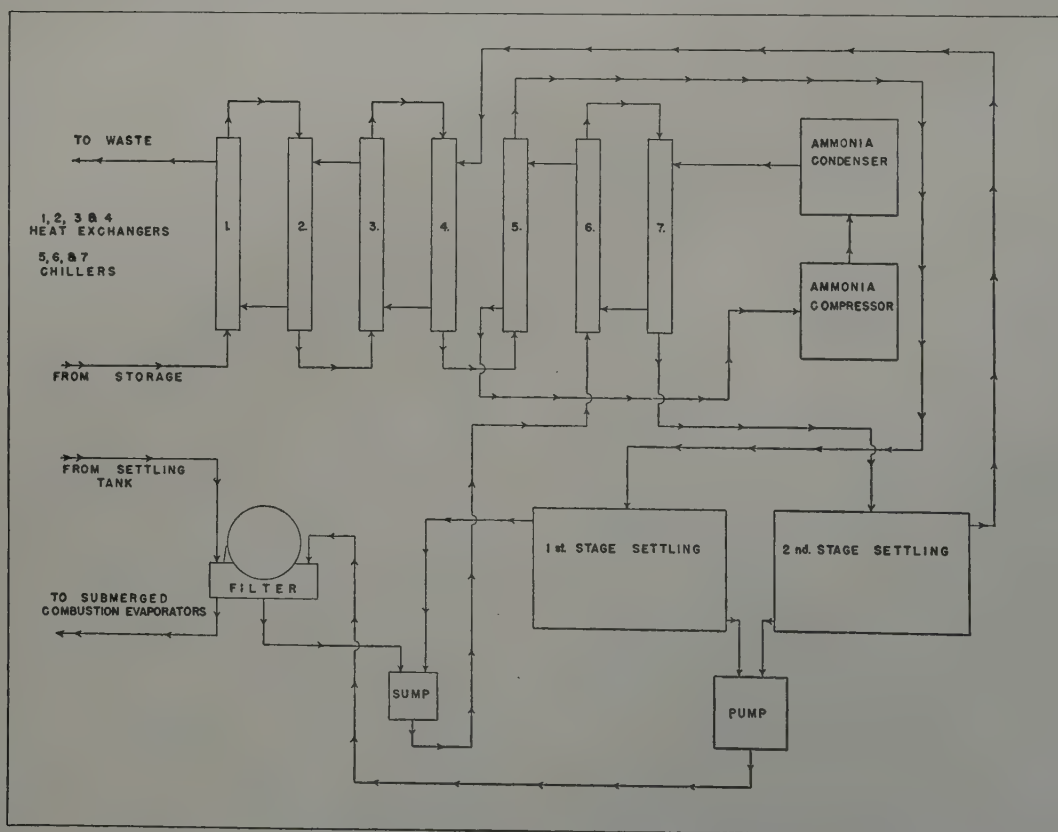


Fig. 2—Glauber's salt crystallization system in chiller plant.

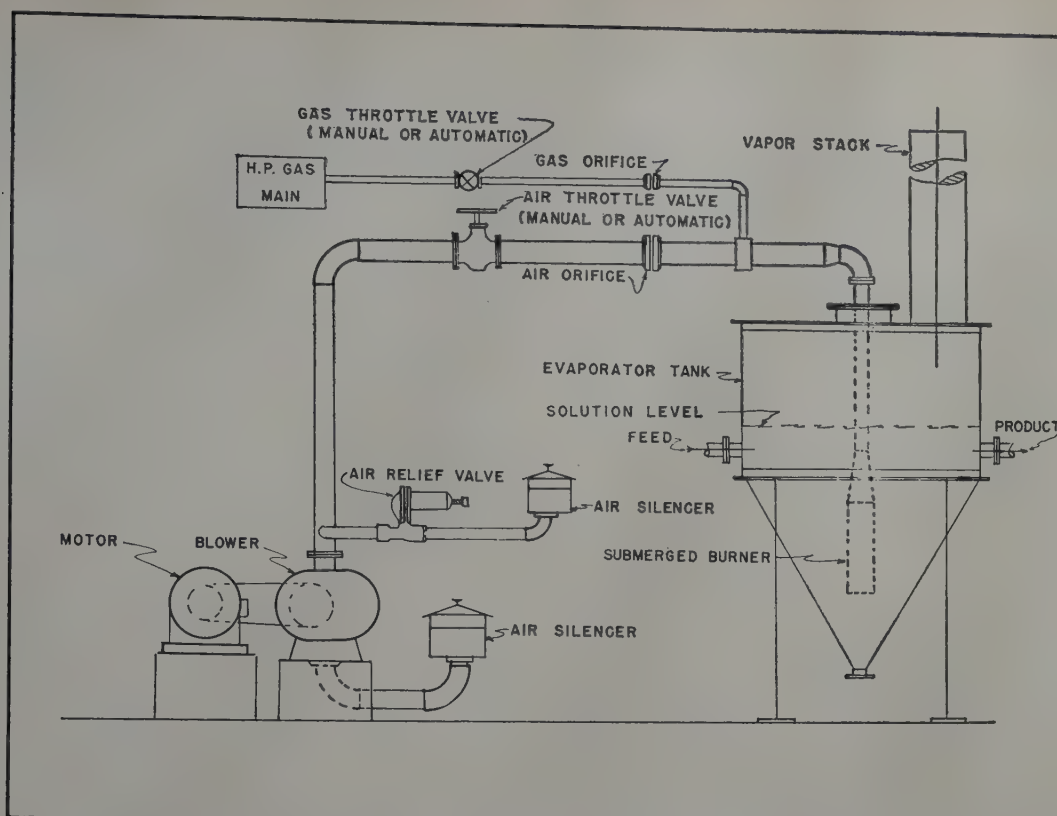


Fig. 3—Schematic sketch of typical submerged combustion installation.

halite, which saturates it with sodium chloride. The halite deposit lies between 1000 and 1500 ft and only one salt well is used, the brine being introduced down the annular space between the tubing and well casing and a chloride saturated brine being pumped up through the tubing.

Fig. 1 is a flowsheet illustrating the handling of the brine. The combined flow from the wells and the lake, or solely from the wells, as the case may be, passes through a weir box which provides a rough approximation of flow and gives the pumper some warning if any of the wells have stopped pumping. From the weir box the brine goes to a settling tank where insoluble material settles out. The overflow goes to a storage tank. Both are wooden tanks 30 ft in diam by 8 ft high. From the storage tank brine is drawn continuously for pumping through the salt well and returned to the system via the weir box. Plant feed brine is withdrawn continuously from the storage tank which serves as a surge tank in case of interruption of flow from the wells.

The plant feed brine is maintained at 1.22 sp gr and contains about 11 to 12 pct sodium sulphate and 12 to 15 pct sodium chloride. The percentage of sodium chloride in the brine has been increased substantially as the original brine contained 9 to 10 pct sodium chloride. In a typical day's operation 43 tons of sodium chloride will be removed from the salt well, but none of it is recovered; all of it after serving its purpose of decreasing the solubility of the Glauber's salt is discarded in the tailing liquor. The plant has a rated capacity of 100 tons of anhydrous sodium sulphate per 24-hr day, and the corresponding flow of brine to the plant is 150 gal per min. Recovery is about 1 lb sodium sulphate per gal of brine passing through the plant. When brine from the lakes reaches a concentration equaling 1.22 sp gr the use of the salt well is discontinued.

Hydrometers are located at several points within

this system to enable manual control of the concentration of the plant feed brine. Brine strength is not extremely critical and slight variations have no serious effect on plant operations.

Fig. 2 is a flow diagram of the Glauber's salt crystallization system which functions in the chiller plant. About 85 pct of the sodium sulphate in the brine can be recovered as Glauber's salt by cooling the brine from 90° to 15° to 20°F. To cool 150 gpm of brine and to crystallize the Glauber's salt requires substantially 500 tons of refrigeration. About 300 tons are provided by an ammonia refrigeration system and the remaining 200 tons are recovered through heat exchange from the waste liquor before it is discharged. Ammonia compressors are of Worthington Pump Co. manufacture. Three 100-ton units are driven by 200-hp Cooper-Bessemer gas engines and one 285-ton unit is driven by a 400-hp gas engine.

Three ammonia chillers and four heat exchangers or recuperators are in use. Chillers and recuperators built by Worthington Pump Co. are similar to the chillers used in petroleum refineries to dewax lubricating oil. Each unit consists of twelve insulated tubes 34 ft long. There is an 8-in. diam outer tube with a 6-in. tube on the inside. The brine flows through the inner tube, which is equipped with rotating scraper ribbons to prevent any precipitating Glauber's salt from building up and adhering to the wall. Flow of brine is countercurrent to the cold mother liquor in the annular space and the four recuperators operate in series. Ammonia is the refrigerant in the annular space of the chillers. The entering brine, which may be as high as 90°F in summer and as low as 60°F in winter, passes through the recuperators, and one ammonia chiller is thereby cooled to 35° or 40°F. The brine is then discharged to the first-stage thickening tank, a conventional Dorr thickener of 20-ft diam.

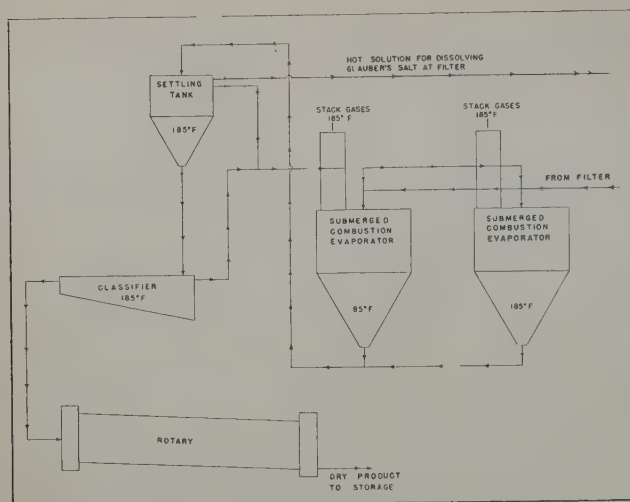


Fig. 4—Flowsheet of drying plant.

The thickened slurry goes to a rotary vacuum filter where the Glauber's salt crystals are filtered and washed. The plant is equipped with both a Swenson and an Oliver filter, but only one is used at a time.

The filtrate and washwater from the filter are combined with the overflow from the first-stage thickener and thence pass through two more ammonia chillers to cool to 15° or 20°F so that additional Glauber's salt will be crystallized. The slurry goes to the second-stage thickener, which is a 26-ft Dorr. The thickened underflow mixes with the underflow from the first-stage thickener and goes to the filter. The overflow is the waste mother liquor at about 20°F and, as stated, is the heat-exchange medium in the recuperators. It is discharged finally to waste at a temperature of 55° to 75°F, depending on the time of year and temperature of the incoming brine. This warm mother liquor is used to flush out and wash out any parts of the system that may have become salted up.

A question often arises regarding the choice of surface cooling instead of vacuum cooling, which is commonly used in similar crystallization problems. Actually, vacuum cooling is efficient only to a terminal temperature of about 40°F, and it is impossible to cool below 32°F by this method. To achieve a good recovery of Glauber's salt it is necessary to chill to about 20°F. Therefore the system in use was chosen.

The drying plant, where the water of crystallization is removed from the Glauber's salt, has been described in detail;³ however, some time is devoted to it here, as it is probably the first commercial application of submerged combustion evaporation.

Sodium sulphate has an inverted solubility curve, which means less solubility at higher temperatures. For this reason it deposits an extremely adherent scale on heat-transfer surfaces such as the tubes of a vacuum evaporator. Operation of this type of evaporator, in which heat is transferred through a tube, becomes very difficult and inefficient because of the rapid build-up of scale and the consequent loss of heat transfer. As a result there are frequent and expensive shutdowns for cleaning out scale.

When operations began at Monahans in 1933 various schemes were tried to effect the dehydration of the Glauber's salt, but at that time none was satisfactory. Rotary drum driers proved impractical because of the severe scaling. For a time the dehydra-

tion was carried out in a counter-current fired rotary kiln. This operation was difficult and inefficient and was replaced by the first commercial submerged combustion evaporator in 1934. The evaporator removes most of the water with final drying of the salt taking place in the rotary kiln.

In the submerged combustion evaporator a gaseous fuel is burned beneath the surface of the liquid, in this case the solution of sodium sulphate, and the products of combustion bubble up through the solution. The burner is submerged at sufficient depth to assure complete and efficient transfer of heat from the products of combustion to the solution. Scaling cannot interfere with this heat transfer, as the heat flows directly from gas to liquid, and rapid heat transfer is assured because the gas is ejected as many bubbles, presenting a tremendous surface area. Furthermore the driving force, which is the temperature difference, is very great, as the flame temperature is around 2600°F.

Fig. 3 shows a typical submerged combustion evaporator. Combustion air is compressed to a pressure of about 7 psi gage and mixed with the proper quantity of gas at equal or slightly higher pressure. The mixture is introduced to the submerged burner and ignition is initiated by a resistance wire heated to incandescence. The flame maintains itself without the presence of any refractory. The controls shown here are manual, as are the controls at Monahans. However, submerged combustion units have been designed with fully automatic control.

The burners used at Monahans are made of mild steel. Some scaling occurs on the tip of the burner, and as it builds up it flakes off, together with a little metal. This action causes sufficient damage to necessitate frequent repair and replacement of burners, but burner cost per ton of production is only a few cents. Some alloys give longer life, but not enough to warrant the extra cost. In other solutions the use of alloys may be desirable, and a graphite burner has been developed for use in acids and corrosive solutions.

Fig. 4 is a flowsheet of the drying plant. About 70 pct of the total water load is evaporated in the two submerged combustion units, or about 110 tons per 24 hr. A dilute slurry of 4 to 5 pct precipitated solids is drawn continuously from the evaporators to a settling tank. The underflow of 20 pct precipitated solids, or 50 pct Na_2SO_4 , is further thickened in an Akins type of classifier to 55 pct precipitated solids or 70 pct Na_2SO_4 . The balance of the water is driven off in the rotary kiln and the dry salt goes to storage for bagging or shipment in bulk.

Instead of thickening the slurry before feeding the salt to the drier, it would be quite feasible to use a centrifuge or filter which could reduce the moisture content of the salt to 4 or 5 pct before drying. Because the submerged combustion evaporator is much more efficient for evaporation of water than the rotary drier, it may seem that substantial savings of fuel could be effected. Actually, however, the price of gas is such that the savings in fuel cost would be offset by the additional power required for a filter or centrifuge.

The overflows from both the settling tank and the classifier are returned to the submerged combustion evaporator, except for a portion from the settling tank which is used to dissolve Glauber's salt at the filter and introduce solution to the evaporators.

The evaporator tanks are made of type 304 stainless steel with highly polished interior surfaces.

After 15 years of service they show no signs of deterioration. Much of the chilling plant has been rehabilitated recently, so that the plant is in a position to meet increased demands now being made on all producers of mineral products.

Each submerged combustion unit burns 170 cu ft of natural gas per min. Combustion air is supplied to each burner by Roots-Connorsville positive pressure blowers driven by 200-hp gas engines.

The plant now has two separate chilling systems, similar to the one described, and interconnected. This means that the same rate of production can be maintained with a weaker brine feed and use of the lake for solar concentration can be eliminated if desirable. If additional production is desired, however, feeding strong brine can increase the production to about 150 tons of anhydrous sodium sulphate per day. Ample filter and evaporation capacity is available in the existing equipment to handle this

tonnage, and present indications are that the plant will be called on for maximum production.

In summary, the Monahans sodium sulphate operation is unique in representing a fusion of several favorable factors that have been fortified with new ingenious developments in processing to yield a top quality product in an industry frequently witnessing failure in wrestling with natural deposits.

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Technical Note

Quantitative Bubble Pick-Up Methods

by S. C. Sun and R. C. Troxell

EFFORTS to obtain definite quantitative data when employing the currently used bubble pick-up method^{1, 2} as a pre-flotation investigation tool led to the adoption of the *magnifying mirror method* and the microscopic method modifications. Quantitative data obtained by these two methods were converted into bubble pick-up indices and bubble pick-up coefficients. Rough correlations were established between the calculated indices and the results of contact angle testing as well as actual flotation under the same conditions.

The basic steps of the magnifying mirror method are as follows: 1—mixing 200 ml distilled water, sized and cleaned mineral particles of approximately 0.5 g, pH regulator, and flotation reagents in a glass beaker; 2—swirling the mineral particles to the center of the beaker to leave a clear circumference; 3—pressing an air bubble against the mineral particles for about 1 sec, and then moving the mineralized air bubble to the clear part of the beaker; and 4—dropping all the attached mineral particles of the air bubble by tapping the bubble holder, and counting the number of the dropped particles by looking at the magnified image in the mirror positioned beneath the beaker. This method is impractical for mineral particles which are transparent in water or finer than 200 mesh.

The microscopic method is essentially similar to the above described magnifying mirror method, with the exception that the mineralized air bubble was drawn up into the bubble holder and subsequently discharged onto an externally placed watch glass, where the number of pick-up particles was counted with the aid of a microscope or hand lens. Drying

was necessary when the discharged liquid interfered with the counting.

The results of the bubble pick-up tests can be simply expressed in the form of the actual count. However, this can be misleading because the number of pick-up particles varies not only with the size of bubble but also with the size of particles. This short-coming is largely remedied by the use of the bubble pick-up index, as calculated from the following equation:

$$PI = (p_n/p) 100 \quad [1]$$

in which PI and p_n are respectively the pick-up index and the number of pick-up particles under any test conditions, and p is the maximum number of pick-up particles of the same mineral under optimum flotation conditions, with the same size holder and particle. The *bubble pick-up coefficient* is a term coined to denote the difference between the bubble pick-up index of mineral particles in a liquid and that of the same mineral-liquid mixture plus a chemical reagent at the same pH value. For example, the difference between the pick-up index of a naturally floatable bituminous coal in a solution of water and kerosene and that of the same coal in water alone is a bubble pick-up coefficient.

Compared with the contact angle method, the quantitative bubble pick-up method is inferior in accuracy but superior in sensitivity. Other advantages of the bubble pick-up method are that the apparatus can be easily assembled, the mineral sample is conventionally prepared, and the test procedure is relatively simple.

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TN 166B. Manuscript, March 6, 1953.

The Blending of Western Coals for Production Of Metallurgical Coke

by John D. Price

COAL blending, in the preparation of coal before coke making, is so commonly practiced as to be almost universal. But the reasons underlying this practice, the benefits resulting from it, and the materials used in blending vary widely. This paper will outline the various phases of the subject and present information which may be correlated with work that has been done elsewhere. It will deal entirely with work done on the high-volatile coking coals of the western part of the United States, special emphasis being given to the coals of Colorado and Utah.

A survey¹ of the 86 coke plants in active operation in the United States during 1949 indicates that only 9 plants, or 10.5 pct of the total, charged one single rank of coal into their ovens, while the remaining 89.5 pct made use of blending in some form. This report indicated that of these total plants 5 used straight high-volatile coal, 4 used straight medium-volatile coal, 47 used blends of high and low-volatile coals, 25 used blends of high, medium, and low-volatile coals, 2 used blends of high and medium-volatile coals, 3 used blends of medium and low-volatile coals.

The fact that certain plants operated on a single kind of coal should not be interpreted to mean that no blending was practiced there, for invariably such plants secure their coal from more than one source and in the interest of uniformity do blend the coals as received.

The general term *coal blending* covers two fields, the first of which is the mechanical mixing of a number of coals to secure uniformity. Often it is found necessary to secure coal for coke production from a number of different mines; these coals, though of the same general type or rank, may differ in their chemical composition or in the physical qualities they impart to coke made from them. Again, it is not unusual to find that coal from different sections of the same mine may show variations in quality. Under such conditions it may be necessary, in the interest of a uniform final product, to introduce a system of blending bins, a bedding yard, or other mechanical methods of securing a uniform mixture. Unfortunately this form of blending has received very little attention up to the present time; it has not received the consideration its value merits.

The second type of blending, while also for the purpose of coke improvement, deals more particularly with the use of a blending agent differing in character from the base coal; it is this form of blending that will be discussed here. To consider only the western coals, for blending may be found necessary for other reasons with other coals, blending has been practiced experimentally or commercially under the following conditions:

1—When a single coal or mixture of coals of the

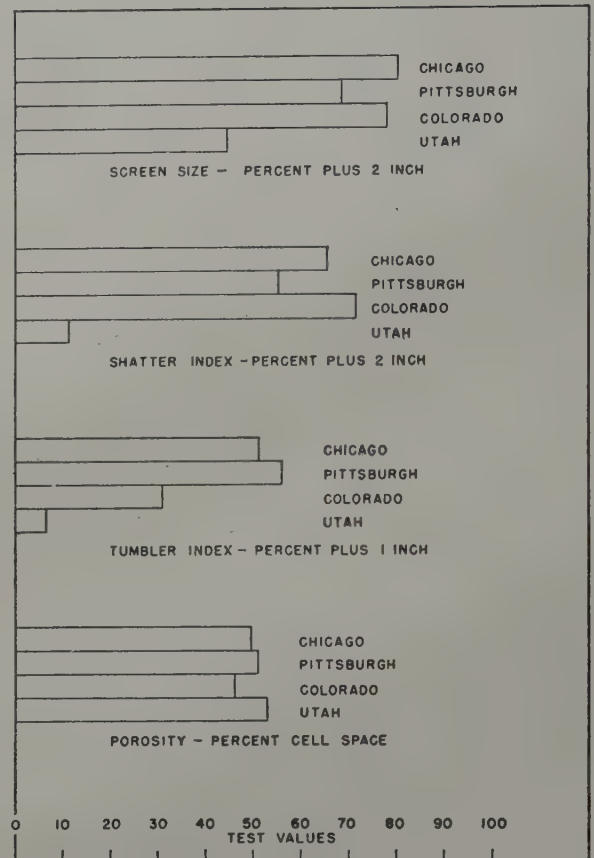


Fig. 1—Comparison of eastern and western cokes.

same rank does not produce a satisfactory coke. For example, a high-volatile coal when used alone is likely to contract when coked so that a comparatively weak coke is formed. Or, if of very low rank, the coal may be deficient in the necessary bitumens required for good coke production.

2—When a product of some special quality is required, for example, when a plant ordinarily producing blast furnace coke must operate at slow coking rate to produce a high-grade foundry coke. Under this condition the reduced daily production of all products which accompanies slow coking time may be undesirable, and the use of some blending agent to increase the size of coke made at faster coking rates may be necessary.

3—When greater yield of coke or its coproducts is needed. Depending upon economic values of the products it may be found desirable to increase the yield of one or the other.

4—When supply of a particular coal must be used, either to protect reserves of high quality coking coal or to utilize a surplus or inferior product not otherwise usable.

Many materials have been used for blending purposes, the exact agent to be used depending both upon the condition to be corrected and the nature of the base coal. No universal blending agent that can

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Discussion on this paper, TP 3544F, may be sent (2 copies) to AIME before Sept. 30, 1953. Manuscript, Nov. 21, 1952. Los Angeles Meeting, February 1953.

Table I. Analyses of Colorado and New Mexico High-Volatile Coals

Mine	Frederick	Morley	Koehler	Crested Butte	Kebler
Classification	Coking	Coking	Light-Coking	Semi-Coking	Non-Coking
Kind	Washed	Washed	Washed	Washed	Washed
Moisture*	2.6	3.8	3.9	7.3	6.2
Volatile matter**	31.0	30.0	37.3	39.6	37.6
Fixed carbon	57.5	55.7	48.4	55.4	51.7
Ash	11.5	14.3	14.3	5.0	10.7
Phosphorus	0.039	0.001	0.001	0.014	0.001
Carbon†	84.61	83.72	82.09	79.78	77.62
Hydrogen	5.74	5.80	6.00	5.63	5.41
Oxygen	7.32	7.86	9.44	12.46	14.83
Nitrogen	1.80	1.80	1.67	1.68	1.48
Sulphur	0.53	0.82	0.80	0.45	0.66
Free swelling index	8.5	7.0	3.5	1.5	0.0
Agglut. index 10 to 1	12.0	10.1	7.7		
Agglut. index 15 to 1	7.7‡	5.8‡	3.4‡		
Source of information	Ref. 8	Ref. 8	Ref. 8	Ref. 8	Ref. 8

* As-received basis, on raw coal from mine

** Moisture-free basis

† Moisture and ash-free basis

‡ Estimated value

be used to improve the coke made from all coals is available. The nearest approach to this will be found in the use of a low or medium-volatile coal as practiced at 75 pct of the plants in the United States. But even in this event each plant must work out the proper type and quality of coal to use and must consider carefully the economics of the practice. In effect, coal blending is an individual plant problem and must be worked out by each plant to secure best results.

At the western coke plants, the following blending agents have been used. Of these the first three have been used on a commercial scale and the last three only in more or less extensive experimental tests: 1—coking coals of lower volatile content and of different expansion properties, 2—non-coking high volatile coals, 3—coal tar pitch, 325° to 350°F melting point, 4—low temperature char, 5—coke breeze, 6—petroleum coke breeze.

Substitute Materials and Procedures

At times it may be found that the blending agent which appears to be most desirable to correct a certain condition is not available in that particular locality and can be secured only at excessive cost. In such event a substitute material can often be secured or produced. Thus while anthracite fines are desirable for use in increasing the size of coke pieces, fine coke breeze can be used in its place with nearly as good results. Again, low-volatile coking coals are

not always economically available, and in such localities low temperature char produced from local high-volatile coals presents a very acceptable alternate. The use of such materials will be touched upon later.

While in most cases not as effective as the use of proper blending agents, certain alternate procedures have been practiced at many plants with gratifying results. These will be merely listed here:

1—Control of carbonizing temperature has an effect on yields of coke and coproducts and on the physical properties of the coke^{2,3} (p. 217 of ref. 2).

2—Degree of pulverization has an effect on the physical structure of the coke^{2,4} (p. 214 of ref. 2).

3—Control of bulk density of the charge has an effect on the expansion pressure exerted by the coal and on the quality of the coke² (p. 93 of ref. 2).

4—Control of moisture content of the charge has an effect on bulk density and consequently on the properties of coke^{2,5} (p. 93 of ref. 2 and p. 136 of ref. 5).

5—Washing of the coal prior to coking has in many cases been proved to benefit both the chemical and physical properties of the resultant coke⁶ (p. 32 of ref. 6).

6—The method of charging ovens has been found to be of importance in securing uniformity of bulk density and quality of coke throughout the whole oven⁵ (p. 96 of ref. 5).

Table II. Analysis of Utah High-Volatile Coals

Mine	Sunnyside	Columbia	Book Cliffs	Castlegate	Huntington
Classification	Light Coking	Light Coking	Semi-Coking	Semi-Coking	Non-Coking
Kind	Raw	Raw	Raw	Raw	Raw
Moisture*	5.8	5.3	9.0	4.1	3.4
Volatile matter**	40.8	40.1	39.2	41.6	43.8
Fixed carbon	53.4	54.4	55.2	52.3	49.6
Ash	5.8	5.5	5.6	6.1	6.6
Phosphorus			0.019	0.012	0.001
Carbon†			79.13	79.66	79.56
Hydrogen			5.74	5.76	5.89
Oxygen	8.7	10.0	12.68	12.68	12.63
Nitrogen			1.59	1.52	1.49
Sulphur	1.2	0.8	0.86	0.38	0.43
Free swelling index					
Agglut. index 10 to 1	4.8	4.5	3.0	2.0	1.0
Agglut. index 15 to 1	8.1‡	7.0‡	3.3	2.6	
Agglut. index 15 to 1	3.9	3.0	1.0‡	0.5‡	
Source of information	Ref. 12	Ref. 12	Ref. 13	Ref. 13	Ref. 13

* As-received basis

** Moisture-free basis

† Moisture and ash-free basis

‡ Estimated value

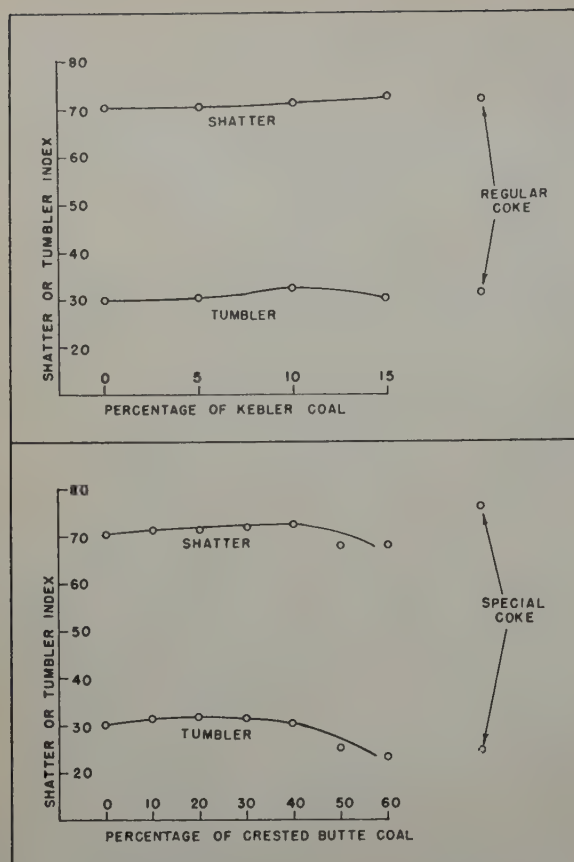


Fig. 2—Effect of blending coals of similar rank.

Very few high-volatile coking coals found in Colorado and Utah can approach the coals of the East as to quality of coke that can be produced from them. In fact, some of these western coals used regularly in the production of coke are so deficient in coking properties that they would be classed as non-coking by an eastern operator. They differ in the main from eastern coals in having been formed in the Cretaceous period and are usually of lower rank and higher oxygen content. Analyses of five representative coals from the Raton Mesa district of Colorado and New Mexico are shown in Table I. Nearly all these coals are high in ash and require washing; the analyses of washed coal samples only are shown. Because Utah coals, on the other hand, are often low

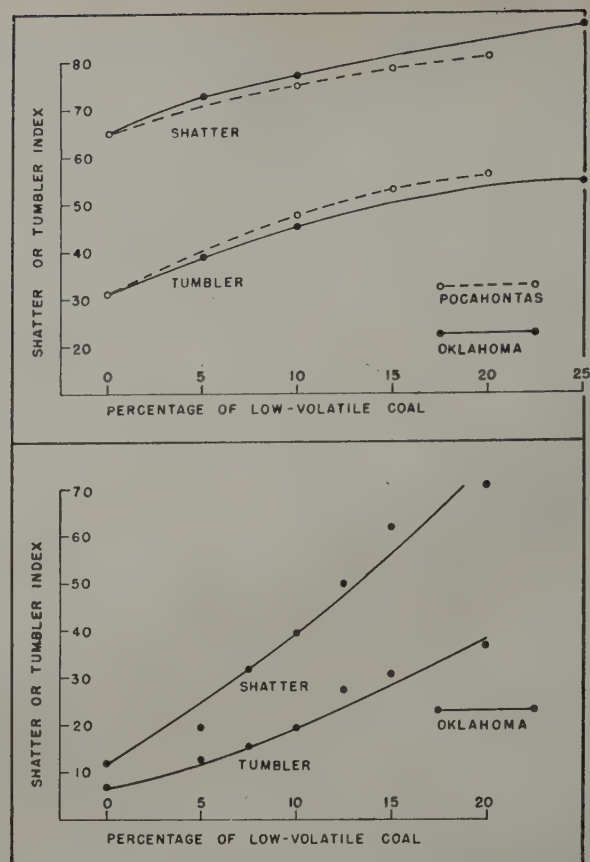


Fig. 3—Effect of blending high and low-volatile coking coals.

in ash, washing is of minor value' (p. 9 of ref. 7). Table I shows analyses of five coals from Utah.

No low-volatile coking coals of a nature similar to Pocahontas are found in the Rocky Mountain states; the nearest source of supply of this kind of coal is in Oklahoma and Arkansas. On the other hand, Kebler coal, a high-volatile, high-oxygen, non-coking coal, the analysis of which has been given in Table I, has been used continuously in small amounts, and char, prepared by low-temperature partial devolatilization of high volatile coal, has been used experimentally. Analyses of such blending agents as have been used with western coals are shown in Table III.

Table III. Analysis of Low and Medium-Volatile Coals and Char

Mine	Carswell*	Philpott	Bokoshe	Stigler	Char
Classification	Low-volatile coking	Low-volatile coking	Low-volatile coking	Medium-volatile coking	
State	West Virginia	Arkansas	Oklahoma	Oklahoma	Colorado
Kind	Raw	Raw	Raw	Raw	
Moisture**		5.3	2.8		
Volatile matter†	17.1	15.9	18.3	24.6	17.4 to 19.1
Fixed carbon	78.0	77.4	66.8	71.1	72.1 to 70.4
Ash	4.9	6.7	14.9	4.3	10.5
Phosphorus		0.014		0.007	
Carbon‡	91.2	89.33	90.9	87.43	
Hydrogen	4.6	4.58	4.4	5.32	
Oxygen	2.1	2.39	2.0	4.38	
Nitrogen	1.3	2.11	1.8	2.13	
Sulphur	0.8	0.93	0.7	0.74	0.6
Free swelling index					
Agglut. index 10 to 1	9.0+	9.0+		9.0+	0.0
Agglut. index 15 to 1	20.0+	20.0+		17.7	0.0
Source of information	12.0§	12.0§		10.0§	0.0
	Ref. 5	Ref. 8	Ref. 8	Ref. 8	Ref. 11

* Pocahontas field

** As-received basis

† Moisture-free basis

‡ Moisture and ash-free basis

§ Estimated value

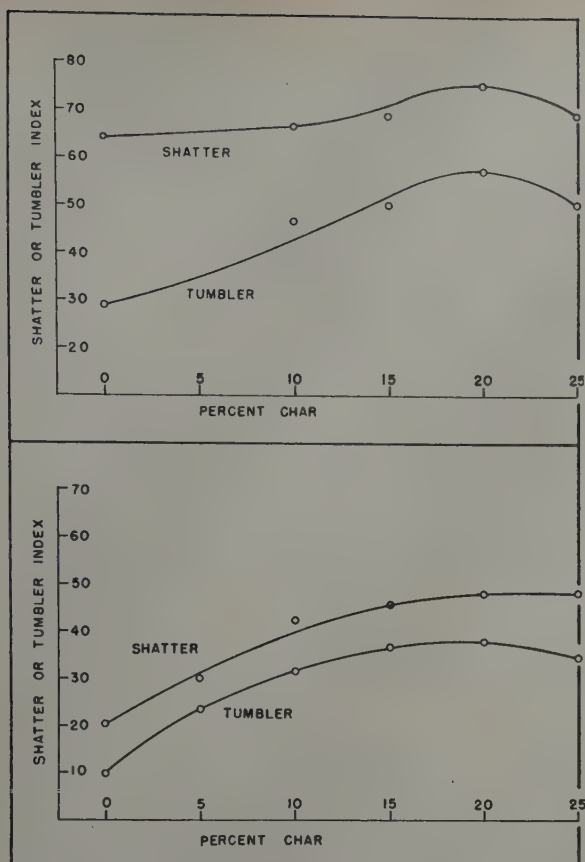


Fig. 4—Effect of use of char.

A comparison of cokes made from eastern and western coals is of value. This is given in Fig. 1, where the various physical properties from representative samples of coke from the Chicago and Pittsburgh districts⁵ (p. 132 of ref. 5) are shown for comparison with similar data on cokes made of coals from Colorado⁸ and Utah⁷ (p. 18 of ref. 7). It must be pointed out here that this comparison is intended to emphasize the need for blending with the western coals, for the eastern cokes were made from coal mixtures containing 20 pct Pocahontas, while coke from western coals was made from straight high-volatile coals. Since the need for blending has been established, and since the examples given here have shown the greatest need for improvement to exist in the shatter and tumbler values, these indices only will be used in a discussion of benefits. The tests should prove to be acceptable criteria of the physical properties of coke, since their cumulative value has been shown to be 80 pct of the total value of coke from a physical property viewpoint at the blast furnaces.⁹

Blends of Colorado High-Volatile Coking Coal with Other High-Volatile Coals of Lower Rank: As shown in Table I, a large number of high-volatile coals of different ranks and coking properties are found in Colorado and New Mexico. Of these Frederick is the only one which possesses properties equal to those of the better coking coals of the East. This coal also has the best washability of any of the coking coals, but alone makes a very dense and fingery coke. Two high-oxygen coals are also found in Colorado, Crested Butte having only slight coking properties and Kebler having none. Between Frederick on the one extreme and these high oxygen coals on the other are found a large number of coals

of various intermediate oxygen content and coking properties. Since both Crested Butte and Kebler coals are desirable because of their ash content, an effort was made to utilize them in the coking mixture; the results of such tests are shown in Fig. 2⁵ (pp. 20-25 of ref. 5). It will be noted that some slight improvement in both shatter and tumbler indices are shown up to 10 pct for Kebler and 30 pct for Crested Butte. Although not shown in Fig. 2, noticeable improvement resulted in the size of the coke prisms from the use of either of these high oxygen coals.⁸ Since the high cost of Crested Butte coal made its use prohibitive, the use of 10 pct Kebler was adopted as standard practice, and since Frederick coal was not available in sufficient quantity, the use of coals similar to Morley and Koehler, see Table I, was necessary. The physical properties of coke made from a blend of 40 pct Frederick, 50 pct miscellaneous coking coals, and 10 pct Kebler is shown as *Regular Coke*.⁸ A *Special Coke* mixture, containing 60 pct Frederick and 40 pct Crested Butte was produced occasionally to make a coke of comparatively low ash content; the physical properties of such coke are also indicated in Fig. 2.

Blends of Colorado and Utah High-Volatile Coking Coals with Low-Volatile Coking Coals: Since the standard blend used in 75 pct of the coke plants in this country consists of various proportions of high- and low-volatile coking coals,¹ it seems reasonable to expect that such blending practice should prove to be of value when applied to the western coals. The effect of this is shown in Fig. 3. In the upper section of this chart it will be noted that Colorado high-volatile coal blended with either Pocahontas or Oklahoma low-volatile coal gave similar results.⁶ Utah coal when blended with Oklahoma low-volatile coal also shows gratifying improvement⁷ (p. 18 of

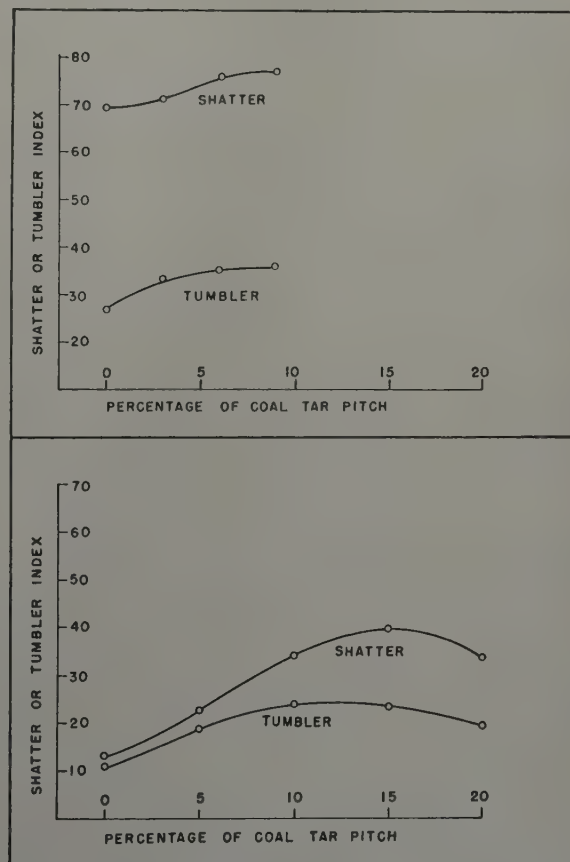


Fig. 5—Effect of use of coal tar pitch.

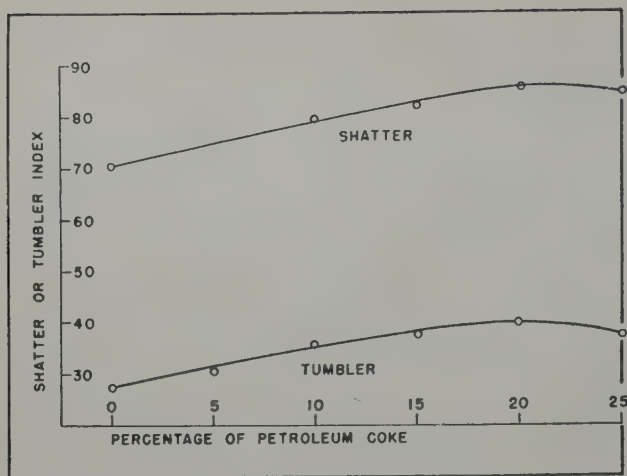


Fig. 6—Effect of use of petroleum coke.

ref. 7) and the result of such blending is shown in the lower section of Fig. 3. One important difference should be noted: with Colorado coal the curves flatten out as the percentage of low-volatile coal is increased, whereas with Utah coal the curve continues to rise. This may be taken as an indication of the deficiency of coke-forming bitumens in Utah as compared with Colorado coals. One additional point is of interest and also serves to illustrate the difference between the Colorado and the Utah coals: the shatter and tumbler values as secured with about 20 pct low-volatile coal additions to Utah coal show physical test values very little higher than those for Colorado coal with no low-volatile coal added.

Blending High-Volatile Coals with Char: It may be found that low-volatile coking coals are not available for economical use as blending agents. In such cases char may be found to be a very desirable substitute. This has been demonstrated at several plants using western coals, and Colorado and Utah coals respond well to the use of this material. In the preparation of char by the partial devolatilization of high-volatile coals at low temperatures, it has been demonstrated that the char must be in effect *tailor-made* to yield optimum results with each base coal used. It has been found that in preparing char the following are important:¹⁰ 1—the coal from which the char is made, 2—the amount of char to be used in the final mixture, 3—the degree of pulverization of the char, 4—the average volatile content of the char and its variation from this average, and 5—the physical properties of the char, with particular reference to its surface.

When char is properly made and used, its effects are similar to those achieved by the use of low-volatile coals. Results obtained from tests of char used with Colorado coal¹¹ (p. 7 of ref. 11) and with Utah coal⁷ (p. 30 of ref. 7) are shown in Fig. 4.

Blending with Coal Tar Pitch: If, as has been previously suggested, Utah coals are deficient in coking bitumens and hence lack fluidity, it would appear that the addition of some bituminous material of highly fluid nature when hot should be of benefit. Coal tar pitch represents such a material, and the effects of its use with Colorado coal⁵ (p. 95 of ref. 5) and with Utah coal⁷ (p. 14 of ref. 7) are shown in Fig. 5. It will be noted that considerable improvement is indicated in both tumbler and shatter indices. One possible reason for this in connection with the Colorado coals may be found in the fact that certain of the coals from New Mexico which are

used in the base high-volatile mixture may be as deficient in bitumens as are the Utah coals.

Blending with Petroleum Coke: The effect of using petroleum coke of about 10 pct volatile matter content in various proportions with Colorado high-volatile coals is shown in the upper section of Fig. 6⁵ (p. 92 of ref. 5). A comparison of these curves with those shown for the use of coal tar pitch will indicate considerable similarity. Both these blending agents were considered impractical because of their scarcity.

Blending with Anthracite Fines or Coke Breeze: The use of specially prepared anthracite fines is not uncommon in cases where it is desired to make a coke of size suitable for foundry use yet retain the volume of products which result from fast coking times. Anthracite fines of the proper screen size specifications are not available in the Rocky Mountain states, and fine coke breeze was used as a substitute. It was found that the desired increase in size could be secured and that this was accompanied by an improvement in shatter values, but also by a lowering of resistance to abrasion. The use of breeze also appeared to affect certain combustibility characteristics of the coke to an unsatisfactory extent, and its use was discontinued before any authentic data were secured.

Conclusion

It should be stressed that each coal presents a problem for individual study; no general rule applies to all. Although it has not been mentioned, the matter of cost must not be overlooked; the cost of blending procedures must be measured against the value of benefits secured. Such a study might disclose that the cost of blending is greater than the value of the improvement in coke quality.

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Mining and Concentration of Ilmenite And Associated Minerals at Trail Ridge, Fla.

by J. H. Carpenter, J. C. Detweiler, J. L. Gillson, E. C. Weichel, Jr.,
and J. P. Wood

ILMENITE, rutile, zircon, and other heavy minerals are found in small amounts in the sand that covers most of Florida. Small enriched deposits occur on the beaches and coastal dunes. One beach deposit east of Jacksonville was worked during World War I and another near Vero Beach during World War II. Both ceased operation when the resumption of imports reduced prices.

Early in World War II reduction of rutile imports made a domestic source of the mineral necessary to maintain the expansion of armaments. Explorations made near Jacksonville discovered a relatively large body of low-grade ore about 10 miles inland from the beaches. A plant installed by Rutile Mining Co. of Florida using both flotation and tables encountered difficulties because of low plant capacity and organic slimes in the ore. In 1944 Humphreys Gold Corp. solved the problem of concentrating the low-grade ore by using the newly developed Humphreys spiral concentrator and by mining with a hydraulic dredge. This combination of low cost mining and concentration permitted the plant to continue operation in competition with foreign ores when imports were resumed following the war.

The success of the Jacksonville operation renewed

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Discussion on this paper, TP 3567AH, may be sent (2 copies) to AIME before Oct. 31, 1953. Manuscript, Feb. 12, 1952. Revised April 26, 1953. New York Meeting, February 1952.

interest in Florida as a source of titanium. An extensive survey of the state was made by E. I. du Pont de Nemours and Co., Inc., under the direction of J. L. Gillson in 1945 to 1946. Sampling was carried on by drive pipe drilling to depths of 20 to 25 ft. Encouraging values were discovered near the location of the present Trail Ridge mine, 50 miles southwest of Jacksonville, near the town of Starke.

The United States Bureau of Mines was interested in exploring for heavy minerals in Florida, and on the recommendation of both the state geologist of Florida and of Du Pont geologists, drilled deeper where marginal indications were found on Trail Ridge. In 1947 it was discovered that sufficiently higher values occurred at depth to make the deposit commercial.

The orebody at Trail Ridge is a long sand deposit averaging about 1 mile in width and 35 ft in depth. It is approximately 1½ miles wide at the north end of the portion now being worked, tapering to a blunt point 3 miles to the south. The ore reaches a maximum depth of 65 ft near the center and feathers out into barren sand at the edges. The ground surface is relatively flat and is covered with vegetation including pine, scrub oak, palmetto, and occasionally in swampy areas, cypress.

The remain of a dense woody growth underlies and marks the bottom of the ore zone. It is difficult to drill through this layer, but in some cases where it is less than 5 ft thick it has been possible to drive through the wood. Sand lying below is coarser than that in the ore zone and contains no values.

The wood layer consists of unconsolidated carbonized fragments of branches and other forest trash associated with a considerable amount of ar-

gillaceous material. Fragments of branches up to 2 in. diam and pine cones have been recovered.

All of the sand in the area is mixed with varying amounts of partially carbonized organic material that contains 70 pct volatile matter and has a clay-like ash. The quantity and consistency of the organic matter varies from a small amount of brown dust at the surface to a black, tar-like material sufficient to cement the sand below the water table into a formation known locally as *hardpan*. The cemented sand occurs in layers and lenses varying in thickness from a few inches to 40 ft with no horizontal continuity. Its hardness is variable, at times resembling soft sandstone.

Values begin at the surface. There are local horizontal and vertical variations in grade, but as a whole 4 pct of the ore is made up of minerals having a sp gr greater than 2.9. Forty-five percent of these are titaniferous. The major associated minerals are zircon, staurolite, kyanite, sillimanite, and tourmaline. Andalusite, pyroxene, spinel, corundum, and monazite occur as very minor constituents.

The principal mineral of the titaniferous group is ilmenite, or more properly its ferric iron analogue similar in composition to arizonite, which analyzes about 63 pct TiO_2 . The balance of the group is a non-magnetic fraction containing altered ilmenite, leucoxene, and a small amount of rutile. The sand as a whole contains about 1.8 pct titanium mineral.

J. L. Gillson¹ has described elsewhere the occurrences and steps in geological history which are common to all commercial deposits of titaniferous sand. These steps are:

1—The existence of a *hinterland* of crystalline rocks in which ilmenite and rutile were accessory minerals, as in the case of the Trail Ridge deposit, Southern Appalachia.

2—A period in which a deep soil zone formed, from which all magnetite and all soluble silicate minerals were leached by organic acids, leaving the quartz, titaniferous minerals, and insoluble silicates behind.

3—An uplift followed by rapid erosion of the soil zone and the dumping of the sands into the ocean, and a later uplift raising the sands to form a source of heavy minerals that again washed into coastal waters.

4—Transportation by ocean currents of the re-deposited sand along the bottom of the sea and the building of offshore bars. In some locations the bars became attached to the shore to form a spit. As these bars or spits grew in height, breakers formed and the force of the waves carried the sand into deep water on the landward side, leaving the heavy minerals behind. Thus a sand deposit was built carrying a modest enrichment of heavy mineral. In India and Brazil slow emergence of low-grade zones and further wave action formed very rich beach deposits. In Florida the elevation was apparently so rapid that there was not time for further concentration of any extent to take place.

Cooke² has shown that during the Sunderland Stage of the Pleistocene Period a small spit was extended from Southern Georgia to the middle of what is now Florida. Sea level at that time was some 200 ft higher than now. It was during the period of this spit formation that local concentration of minerals took place and an orebody was formed.

Later, when the spit emerged, it became the backbone of peninsular Florida and is now called Trail Ridge. Discovery of an area in which the titanium

minerals were concentrated in sufficient amounts to be recovered by modern mining and milling methods led to the development of the Trail Ridge mine.

Drilling: The sandy nature of the ore and the fact that the water level was very near the surface precluded the use of ordinary drilling methods to prospect and develop the deposit. Development of a modified core drill by the cooperative efforts of the Bureau of Mines, Du Pont, and Humphreys greatly facilitated the exploration and proving of the Trail Ridge orebody. With this type of drill, the core is continuously removed by a flow of water within the drill casing. The drilling assembly consists of a 200-lb hammer, an elongated hollow drive head with a hose connection in the side, 10-ft lengths of $\frac{3}{4}$ -in. jet pipe, and 1½-in. diamond drilling casing. The first length of jet pipe passes through and is welded to the hammer. The end of the pipe above the hammer is connected by a hose to a pump. Its length below the hammer is such that it extends through the drive head to within 6 in. of the bottom of the first section of casing. Consequently, as equal lengths of jet pipe and casing are added, the relative position of the lower ends of the two pipes is always the same. As the casing is driven, water pumped through the jet pipe breaks up the rising core of sand in the casing and carries it to the drive head where it discharges through the side opening and flows through a hose to the sample tub. Holes 75 ft deep are frequently drilled.

Samples are obtained from each 5-ft section of ore. Drilling is continued until some formation such as clay or wood layer is reached, or when panning of the samples indicates that the bottom of the ore has been passed. A 3-man crew averages about 200 ft of hole per day. In the early development, drill spacing was on 330x660-ft. lines; for operational purposes holes are spaced on 100-ft. centers. The equipment, mounted on a 6x6 truck, includes a folding 25-ft mast, a walking beam, a hoist, a water pump, a gasoline engine, and a 250-gal water tank.

The development drilling by Du Pont indicated that there was a sufficient reserve on the land held by the company to justify the erection of a plant to treat 20,000 long tons of ore per day for the production of 100,000 long tons of titanium mineral a year. In January 1948 Du Pont engaged Humphreys to design, construct, and operate a plant to exploit this reserve.

Dredging was considered the most economical means of mining this type of deposit. The next consideration was the location of the concentrator in relation to the dredge. Alternate plans were proposed: 1—to bring the ore to the mill and 2—to bring the mill to the ore.

The first proposal, to locate a mill centrally on land, required pumping approximately 1100 long tons of sand per hr and 18,000 gal of water per min an average distance of over a mile. The high ratio of concentration and the necessity of backfilling the mined area would require pumping nearly an equal amount of tailing and water the same distance.

According to the second plan, a plant floating on the dredge pond could follow the mining. It would then be necessary to pump the large amounts of feed and tailing less than 1000 ft. Pumping requirements for any considerable distance would be reduced to approximately 40 tons of concentrate per hr with 500 gal of water per min.

The second proposal was adopted and the plant was designed and built. Construction was started in

April 1948 and completed in a year. Hulls for the dredge and concentrator were assembled in dry pits which were later flooded, and the equipment was installed on the floating units. The frontispiece, an aerial view of the plant, shows the dredge and wet mill in the pond at upper right, the acid treatment settling area to the left, and the dry mill, zircon mill, offices, and shops in the center foreground.

The dredge hull was fabricated from plates cut and welded on the site. The three barges carrying the concentrator were assembled from navy pontoons designed for construction of temporary docks and barges.

Floating Concentrator: The plant is unique in its compactness, simplicity of flowsheet, see Fig. 1, and mobility. Three barges with total displacement of 3000 tons and total deck area of 27,000 sq ft carry all the equipment necessary to process 20,000 tons of feed per day. Separate barges were used to provide a space of suitable size and shape for each group of equipment.

One barge, 100x112 ft, carries two 6x12-ft vibrating screens, an oversize crusher, seven 12x30-ft dewatering classifiers, and the spiral feed pumps and bins, see Fig. 2. The 72x126-ft rougher spiral barge, Fig. 3, supports 704 spirals, product transfer pumps, the main tailing bin, and tailing pumps. The 62x109-ft cleaner barge, Fig. 4, carries spirals used in the cleaner and re-cleaner circuits, the cleaner feed bins, and pumps for feed, tailing, and concentrate. The plant uses a total of 30 pumps, ranging in size from 3 to 16 in.

The barges are tied together by flexible connections, which allow for vertical movement as the load varies on each section and permit the plant to be moved as a unit. Interbarge pipelines are connected with dredge sleeves and hose.

Power totaling 1500 kw at 2300 v is carried to the barges on 3-conductor cables. Feed from the dredge reaches the barges through a pipeline floated on pontoons. Tailing pumps are connected through dredge sleeves to shore lines which transport tailings to the area being backfilled.

The plant is normally tied to the bank of the dredge pond, but at times it is moored in open water and held by cables attached to deadmen on shore. At the present mining rate the plant is moved about every four weeks. Movement of the plant requires

approximately 12 hr including rearrangement of shore pipelines and power cables.

Operations

Clearing: As there is no overburden to be removed, it is necessary to give considerable attention to the removal of surface vegetation to prevent fouling of the dredge cutterhead and pump.

After standing timber is removed, stumps are pulled with a stumper and large roots are picked up with a root rake mounted on the front of crawler tractors. A roter and a tandem disc, which are pulled by a tractor, unearth and reduce the smaller roots to short lengths that will pass through the dredge pump. Trash, stumps, and roots are gathered and burned.

This clearing operation removes the majority of the roots. Those remaining, combined with fragments of stumps which occur 5 to 10 ft below the surface, originally gave trouble in the dredge pump. The problem was solved by installation of a plant-developed *root hog* and *wiper blade* described below.

Blasting: The presence of hardpan and the poor flow characteristics of the sand make it necessary to blast ahead of the dredge. Blast holes are drilled in rows about 20 ft apart and parallel to the face of the dredge pond. The drill rig developed at the plant has a drill mounted on each end of a half-track. This arrangement permits two holes to be drilled simultaneously at 28-ft intervals in the rows. Each end of the blast drill rig is equipped with a 45-ft mast and an electrically driven capstan. A 40-ft drill pipe with a 6-in. diam bladed cutter is supported by a rope passing over the masthead pulley to the capstan. High pressure water is carried to the top of the drill pipe by a hose. No hammer or casing is used. Alternate tightening and loosening of the rope, which is wrapped around the capstan head, imparts a chopping action to the cutter blade while water is pumped into the hole.

Sand, loosened by the cutter blade and the jet of water, is flushed out and leaves an open hole of about 8-in. diam extending to the bottom of the ore. Holes are loaded with DuPont Nitramon and detonated with Primacord and millisecond delays.

Dredging: The suction cutterhead dredge, Fig. 5, is capable of digging 1200 tons per hr to a depth of 45 ft below the water level. A 20-in. dredge pump powered by a 750 hp motor is used as a means of

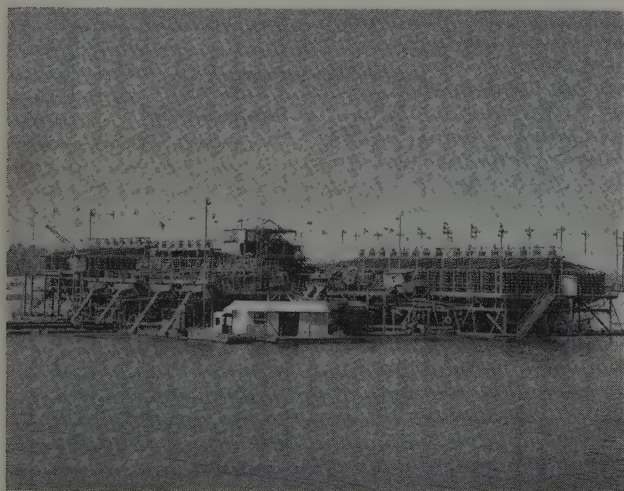


Fig. 2—Floating concentrator. Screening and dewatering equipment is in the center, the rougher spirals to the right, and finisher section to the left.

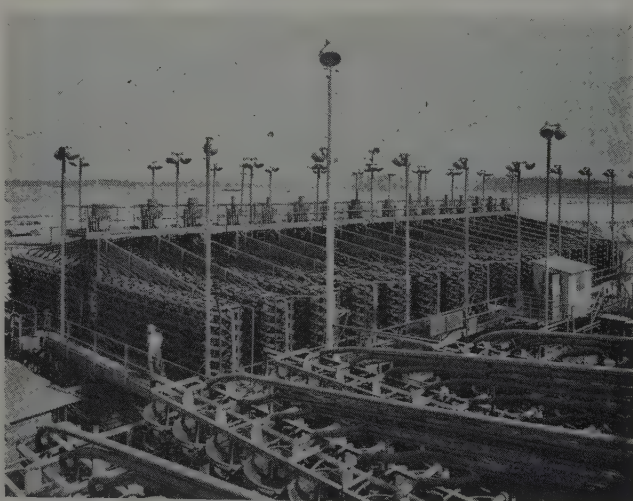


Fig. 3—A view of the 704 7-turn rougher spirals, showing the feed system. The barge which supports this equipment is 72x126 ft.

moving the ore from the ground to the wet mill. The dredge pump suction line, of 24-in. diam, is extended down the ladder and terminates inside the cutterhead. The revolving cutter loosens the sand so that it may be picked up by the pump suction.

The cutterhead is made of six curved blades assembled to form a hemisphere. The blades are shaped in such a way that when the cutterhead revolves, sand is forced toward the pump suction intake within the cutterhead. The head is attached to a 9-in. shaft driven at 16 rpm by a 200-hp motor through a gear reducer located near the top of the 65-ft ladder. Digging depth is controlled by raising and lowering the ladder.

The dredge pump discharges to a 24-in. pipeline, made up of 40-ft sections of pipe supported on pontoons, which connects with the wet mill. Connections between joints are made with rubber dredge sleeves. The flexibility of the couplings, together with the arrangement of the line in an ell shape, permits pipe to be added or taken out in sections ranging from 80 to 160 ft. The line may vary 280 to 1060 ft, depending on the dredge position.

Power totaling 800 kw at 2300 v is conducted from shore to the dredge by a 3-conductor cable supported on pontoons.

When mining, the dredge is stepped forward into the toe of the bank by the alternate use of two spuds located at the stern. Using one spud as a pivot, the dredge is swung from side to side so that the cutterhead makes a cut approximately 200 ft wide. The swinging is controlled by swing winches on the dredge which haul on lines attached to 2½-cu yd drag-line buckets acting as deadmen on shore. Drag-line buckets are used because of their tendency to pull into the ground when tension is applied to the lines. The dredge starts a mining cycle about 200 ft from the wet mill and moves forward, in successive cuts about 150 ft wide, to the full extent of the pipeline. Usually the area mined in one cycle is equivalent to 8 acres. The wet mill is then moved forward and the cycle repeated. The dredge operator is guided by markers on shore, a depth gage, and a chart showing the depth of ore at each station.

Because of lack of storage capacity at the wet mill, the dredge must produce at a uniform rate to maintain proper spiral loading. Control of the production rate is aided by vacuum and pressure gages

installed at various points in the pumplines and by an ammeter on the dredge pump motor. A flow meter which integrates the flow and pulp density to dry solids weight gives the operator a continuous chart of dredging rate and total tonnage dredged.

Root Hog: Roots not recovered in clearing are waterlogged and sink to the bottom of the pond. Before the *root hog* was developed these and large chunks of hardpan picked up by the suction severely handicapped operations by plugging the pump.

The root hog is essentially a chopper contained in a box welded into the dredge pump suction line. A punch plate with 5-in holes starts at the bottom of the inlet to the box and slopes upward at 15° towards the discharge, ending in an enlarged chamber which holds the chopper.

The bulk of the material entering the root hog passes through the plate openings and to the pump. Larger pieces of roots and hardpan are carried by the force of the stream to the chopper which acts as a combination hammer mill and wood chipper. It consists of a heavy rotor with teeth that mesh between a fixed set of dies. The rotor turns at 300 rpm, crushing the oversize and roots between the dies and teeth. The reduced matter falls into the main stream and passes through the pump.

Power is applied to the rotating element by a drive outside the box. Shear pins protect the equipment from damage by tramp iron. Stuffing glands around the rotor shaft prevent loss of vacuum in the suction line.

Wiper Blade: The long thin roots that passed through the punch plate had a tendency to hang up on the leading edge of the impeller vanes of the dredge pump. The wiper blade was developed to eliminate the periodic shutdowns required to clean out the pump.

A 1½x4x5-in. steel blade is welded to the suction face plate so that it is parallel to the pump shaft and extends into the pump the full width of the vanes. Clearance between the blade and vanes is about ½ in. As the impeller rotates, small roots hanging on the vanes are sheared off and pass through the pump.

Wet Plant: The wet mill treats approximately 1100 tons of ore per hr containing 4 pct heavy mineral and produces about 45 tons of concentrate per hr, averaging 85 pct heavy mineral. This is accom-

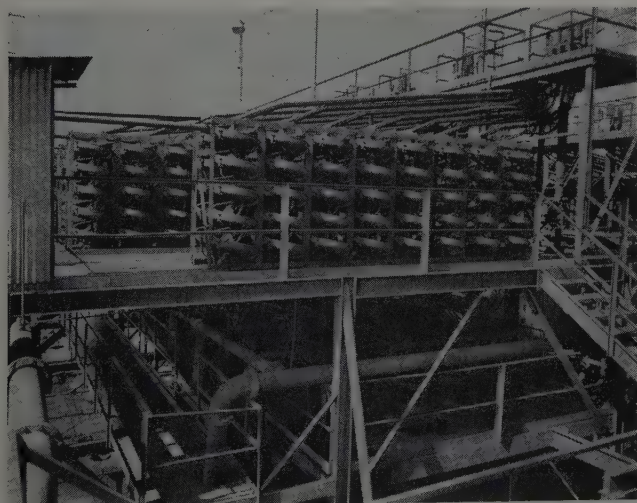


Fig. 4—A portion of the cleaner and re-cleaner section. The 5-turn cleaners are on the upper deck with the 3-turn re-cleaners below.



Fig. 5—Dredge. The 200-hp cutterhead drive is on the forward end. The spuds and the supporting frames are at the stern.

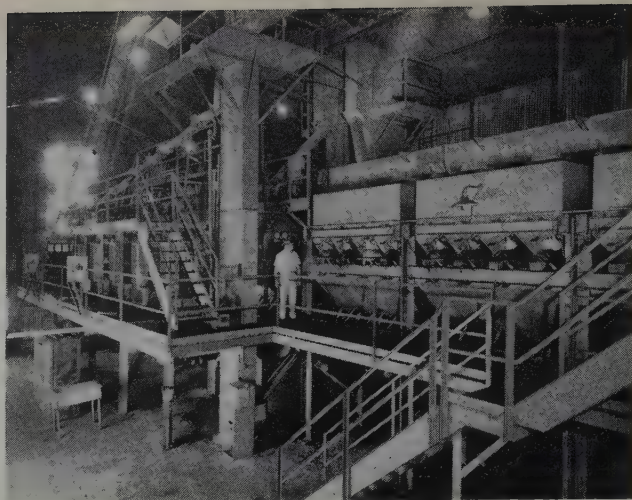


Fig. 6 (left)—Exterior view of the ilmenite dry mill. Fig. 7 (right)—Interior view of the ilmenite dry mill. At the right is a portion of the high tension separators which split the titanium minerals into silicates. At the left are the magnets which extract staurolite from the silicate fractions.

plished in three stages of spiral concentration as shown in Fig. 1. Feed from the dredge is screened at $\frac{1}{4}$ in. on vibrating screens. The oversize is conveyed to a hammer mill to reduce hardpan. Roots and wood chips in the mill discharge are removed on a fixed screen while the freed sand grains are returned to the screens. Undersize from the screens is dewatered in rake classifiers, repulped to approximately 30 pct density and pumped to the rougher spirals. This is the only preparation required for the plant feed. The product called middling in the flowsheet is low-grade concentrate which is recirculated for further concentration with no additional treatment. Each Humphreys spiral is made of cast-iron trough-shaped sections so designed that when assembled they form a helix of the required number of turns. The pulp is introduced at the top and flows by gravity down the spiral. The sand and water are carried to the outer section of the trough. The lighter sand grains and most of the water continue to flow in this section and are discharged as tailing at the bottom. The tailing flows to the main tailing sump and is pumped to the dredged area being backfilled. The heavy mineral particles settle out and migrate to the inner edge of the trough where they form a thin moving band. The mineral that has reached the band during each turn is drawn off through ports connected by hose to a two-compartment collector pipe in the center of the spiral. Concentrate withdrawn from the top turns enters the upper port of the collector pipe, and middling from the lower turns is drawn off to the lower port. The rougher spiral section contains 704 seven-turn spirals rejecting a final tailing which is pumped to spoil areas. The concentrate is retreated in 264 five-turn cleaner spirals. Tailing from this circuit is recirculated through the roughers. The cleaner concentrate is further treated on 132 three-turn spirals to produce finished concentrate and a recirculating tailing. The finished concentrate is pumped to the dry mill where it is dewatered, scrubbed to remove adhering organic slime, and stockpiled for further treatment in the dry mill.

Waste Water Treatment: Pond water is used for make-up and wash water wherever possible. However, it is necessary to use fresh water for pump

seals and wash water on the finisher spirals. This water, along with the accumulation from heavy rainfall, makes it necessary to waste approximately 3000 gal per min to public streams. The organic material in the orebody forms colloidal slime in the pond which has a very high coloring capacity. The discharge of this water to streams without clarification would seriously interfere with natural plant and animal life. The slime is flocculated by reducing the pH of the excess water to approximately 3.5 with sulphuric acid. The water then flows to a 20-acre settling pond left from previous mining operations where the coagulated sludge settles. The clear overflow from the pond is neutralized with lime before entering the stream. Both acid and lime are fed by automatically controlled equipment.

Maintenance: Abrasion of all iron equipment by the sand presented a major maintenance problem in the operation of the wet mill. Rubber and neoprene proved to be the answer to the problem. Approximately $7\frac{1}{2}$ miles of rubber and neoprene tubing and hose, ranging in size from $1\frac{1}{2}$ to 8 in. diam, have been installed as replacement for pipelines and small launders.

Tubing is supported in angle iron racks or used as liners for pipes to carry streams flowing by gravity. High pressure hose is used to replace pumplines whenever possible. The larger launders are lined with $\frac{1}{4}$ -in. rubber sheet bolted in place.

All the spirals have been lined with $\frac{1}{4}$ -in. sheet Neoprene at the plant by a process of dry vulcanizing. This process is also used to line launders, bins, and other equipment subject to sand abrasion. Neoprene-lined launders in service for three years show negligible wear compared to the complete failure of a similar thickness of steel within three or four months. Rubber-lined pumps have proved to be very satisfactory at this plant. Nearly all the sand pumps in use are of this type and replacement of the remaining metal pumps, except the one on the dredge, is scheduled in the near future.

Of particular interest are the newly developed 16-in. rubber-lined pumps believed to be the largest in use at this time. These pumps have a capacity of 8000 gal per min and handle approximately 14,000 tons of sand per day. One has been in use for over six months without appreciable wear.

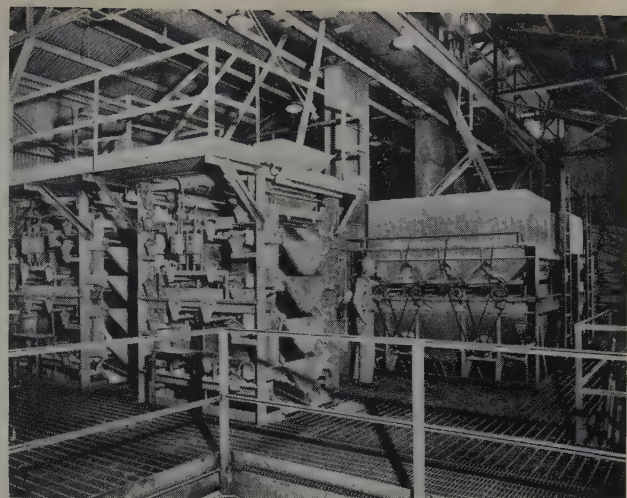
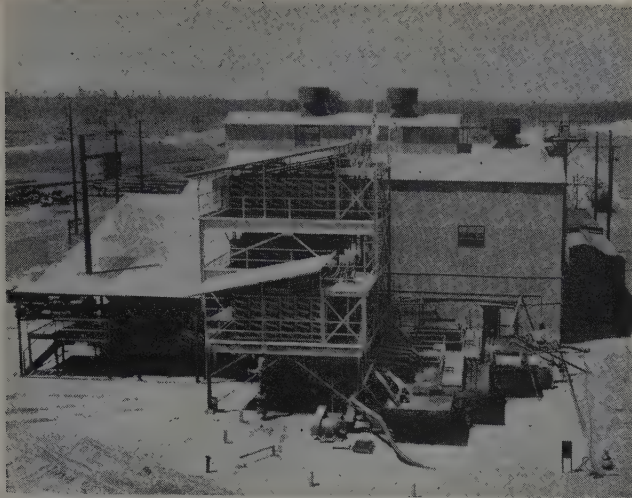


Fig. 8 (left)—Exterior view of the zircon unit. Fig. 9 (right)—Interior of the zircon unit showing high tension separators and magnets. Similar magnets are used in the ilmenite mill to separate ilmenite from the non-magnetic titanium minerals.

Dry Plant: The bulk concentrate is further treated in a dry mill, Fig. 6, having a capacity of 700 tons of feed per day from which 300 tons of titanium mineral concentrate and 120 tons of staurolite is extracted. Two methods of concentration are used: high tension, an improved form of electrostatic separation, and high intensity magnetic separation, see Fig. 7.

The feed is dried and heated to approximately 220°F and transported to the mill by shaking conveyors. It is elevated by bucket elevators to screw conveyors which deliver the feed to bins over each concentrating unit. Horizontal belts underneath the units gather and transfer the various products.

Titanium minerals are separated from the silicate minerals by high tension separators. Sixty-two of these units are arranged in four groups for roughing, scavenging, and cleaning steps as shown in the flowsheet, Fig. 1. In this type of separation the minerals are fed onto a high-speed spinning rotor, and a heavy corona discharge is aimed toward the minerals at the point where they would normally leave the rotor. A 40,000-v pulsating dc is used.

The minerals of relatively poor electrical conductivity are pinned to the rotor by the high surface charge they receive on passing through the high voltage corona. The minerals of relatively high conductivity do not as readily hold this surface charge and so leave the rotor in their normal trajectory. In this particular operation the titanium minerals are the only ones present of relatively high electrical conductivity and are, therefore, thrown off the rotor. The silicates are pinned to the rotor and are removed by a fixed brush.

In the rougher stage, where the minerals are the hottest, the non-conductor minerals are pinned so strongly to the rotor that it is difficult to brush them off. This problem has been overcome by the use of a second electrode with a high voltage alternating current which discharges the grains, allowing them to be removed easily.

Titanium minerals from the high-tension separators are treated by induced roll magnetic separators to produce ilmenite at 63 pct TiO_2 and a non-magnetic fraction, largely leucoxene, which after two-stage high tension cleaning contains about 80

pct TiO_2 . The titanium mineral products which contain less than 2 pct silicates are shipped in hopper-bottom covered cars to E. I. du Pont de Nemours and Co., Inc., Wilmington, Del.

The silicate reject from the high-tension scavengers is further treated on induced roll magnets to produce a magnetic fraction which is largely staurolite but also contains some tourmaline. This product contains more than 45 pct Al_2O_3 and is used by a Florida cement plant as a source of alumina in the mix instead of the customary clay.

Zircon Plant: Humphreys Gold Corp. operates a separate plant, Fig. 8, to recover the zircon contained in the ilmenite-staurolite tailing. The tailing is re-spiralled to remove quartz, kyanite, sillimanite, and other minerals that cannot be separated by dry methods. The spiral plant uses 120 spirals arranged in four stages and produces a concentrate containing about 90 pct zircon and less than 1 pct of either quartz or non-magnetic aluminum silicates. Fig. 9 shows the interior of the zircon plant.

The spiral concentrate is dried and heated to 1200°F in a rotary oil-fired counterflow dryer to remove the small amount of organic staining which discolors the surface of the zircon grains. This calcining results in a snow-white zircon product.

The discharge from the drying-calcining operation is elevated to the dry plant feed bin by an air lift which also acts as a cooler. In the dry plant 16 high tension separators are used to remove residual titaniferous minerals. Further treatment on two induced-roll magnetic separators removes non-conductor magnetic minerals, mainly staurolite and tourmaline. The non-magnetic fraction, 98 pct zircon, is packed in paper bags of 100 lb each. The plant has a capacity of 2500 short tons per month. The zircon product is marketed for foundry, ceramic, and high temperature use, and for special sandblasting.

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National Lead Co. Mechanization At Fredericktown, Mo.

by Harold A. Krueger

FACILITIES and mining operations of the National Lead Co., St. Louis Smelting and Refining Division, near Fredericktown, Mo., are situated in a famous mining area. Copper, lead, nickel, and cobalt have been mined here for more than 100 years, work having been started on a high sulphide copper outcrop in 1847.

Lamotte sandstone is characterized by differential compaction on a rigorously eroded pre-Cambrian surface. The Bonneterre formation was therefore a good host for minerals not generally found in mineable quantities in these midwestern areas. Unusually complex minerals, however, make beneficiation difficult, and because of irregular ore thicknesses and elevations many engineers and operators have not attempted to mine the property. Others have tried who failed.

This paper deals with economic, efficient, and competitive methods of mining these highly irregular orebodies, as compared to the open-stope, room-and-pillar methods normally used for horizontal-bedded lead deposits.

For the purpose of this study it should be understood that the ore is found in two distinctly different types of occurrences, one to be designated as *basin ore* and the other as *contact ore*. Mining of basin ore is complicated by many faults, fractures, cross faults, and breaks. Contact ore is complex because it is found on flanks or slopes of pre-Cambrian knobs or highs. The dip of the mining floor for the latter type varies between 18° and 45°.

Occurrences of both types of ore are complicated by water courses or solution channels which carry unconsolidated shale, lime, sand, and dolomite. This material is also found between the bedding planes of the members of the Bonneterre formation. The water found where there are fractures, faults, and channels makes it very fluid and tacky, see Fig. 1, particularly after it has been blasted and handled by loading and hauling machines. Much of the ore can be wadded and thrown without dispersing.

During early operations by the Buckeye Copper Co. in 1861 and the North American Lead Co. from 1900 to 1910, conventional narrow-gage railroad and side dump mine cars were used with hand shoveling. The complications of mining the contact ore, the only type attempted at this time, can be appreciated when it is realized that operators were obliged to use mules for haulage. Haulageways constructed on these slopes were of necessity similar to wagon trails or goat trails up the side of a mountain. In other words, it was merely a matter of going from side to side of the strike length of the slope, gaining

a little in elevation on each shuttle trip. Production totaled only one to two tons per manshift.

A few years later, about 1913, the property was purchased by combined Canadian interests known as the Missouri Cobalt Co., and the use of trolley locomotives was initiated.

Between 1900 and 1928 a land agent using churn and diamond drilling methods prospected scattered sections of the area. In 1928 the first property was purchased by the present company, then operating as the St. Louis Smelting and Refining Co. Check drilling and prospecting was carried out by the company at various times between 1928 and 1939 to correlate the erratic mineralization. Much information about both types of orebodies was accumulated, but it was still questionable as to whether money should be invested to work these occurrences.

In anticipation of high lead and copper prices, about the time World War II started, it was decided to develop and bring into production some of this ore. In 1942 No. 1 shaft was put down on the largest basin-type orebody and in 1943 No. 2 shaft was put down on contact-type ore. Operations were expanded when No. 3 shaft was completed in 1943, and progressed further in 1948, when National Lead Co. dewatered and opened No. 5 and 6 mines, old workings of the North American Lead Co. and the Missouri Cobalt Co.

Because of the differential compaction of Lamotte sandstone over the pre-Cambrian porphyry, in some instances mineable thicknesses of basin-type ore occurred 20 to 30 ft above the sand. This is the exception rather than the rule, since most of the mineralization starts at the sand and is variable in thickness. The ore was attacked, therefore, by development drifts and crosscuts at the lowest possible elevation, where the ore immediately overlying the Lamotte sandstone could be drained and made accessible for mining. It was planned to connect to the drifts and crosscuts with raises to mine ore deposited 20 to 30 ft higher. The higher orebodies were thus mined as slusher levels. Slusher hoists were used to drag the ore into the raises, which were made into hoppers. The ore was then loaded into 32x32-in. ore cans, hauled to the shaft by battery locomotives, and hoisted by the conventional Tri-State method. The rate of efficiency was 5 to 6 tons per manshift underground.

The contact-type ore was attacked in a similar way, except that the orebodies were not nearly so wide, so that they were more flexible for slusher loading into cans. This advantage was offset, however, by haulage complexities, since the railroad was constructed on steep slopes.

Through experience and ingenuity, many improvements were made in mining both types of ores. The two levels, so-called, in the basin-type orebodies were connected as previously planned, more efficient locomotives replaced the older ones, and a

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Discussion on this paper, TP 3584A, may be sent (2 copies) to AIME before Oct. 31, 1953. Manuscript, April 11, 1953. Los Angeles Meeting, February 1953.

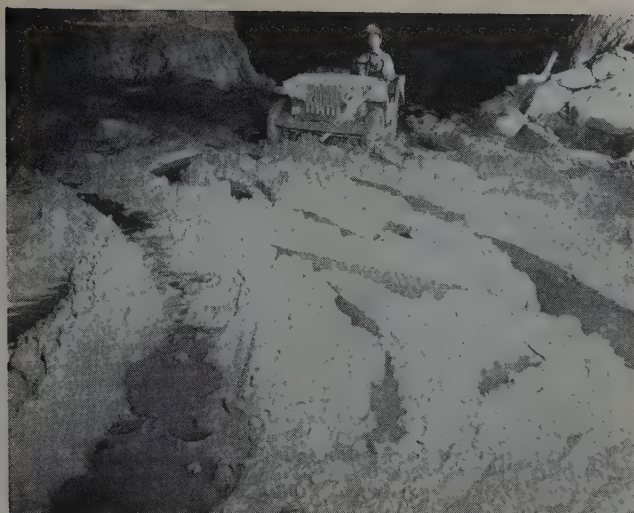


Fig. 1—Mud condition caused by water present where there are fractures, faults, or channels.

better shaft-switching system was used prior to hoisting the cans. In the early stages of mining, the upper and lower ore did not connect in all instances, but where it did, the resulting abrupt change in floor elevation was leveled off by means of railroad fill whenever possible. Improvements in draining the mine of muddy, sandy water also contributed to the steady increase in operating efficiency, which from all indications had reached a maximum, 8 to 10 tons per manshift underground, by the middle of 1949.

With the years 1944 to 1947 used as a based period, the average yearly grade of ore, total mining cost per ton, labor cost per ton, labor rates, materials cost per ton, and material cost and tons per manshift have been plotted on a line graph, Fig. 2. This shows the trends existing during the five-year period from 1947 through 1952.

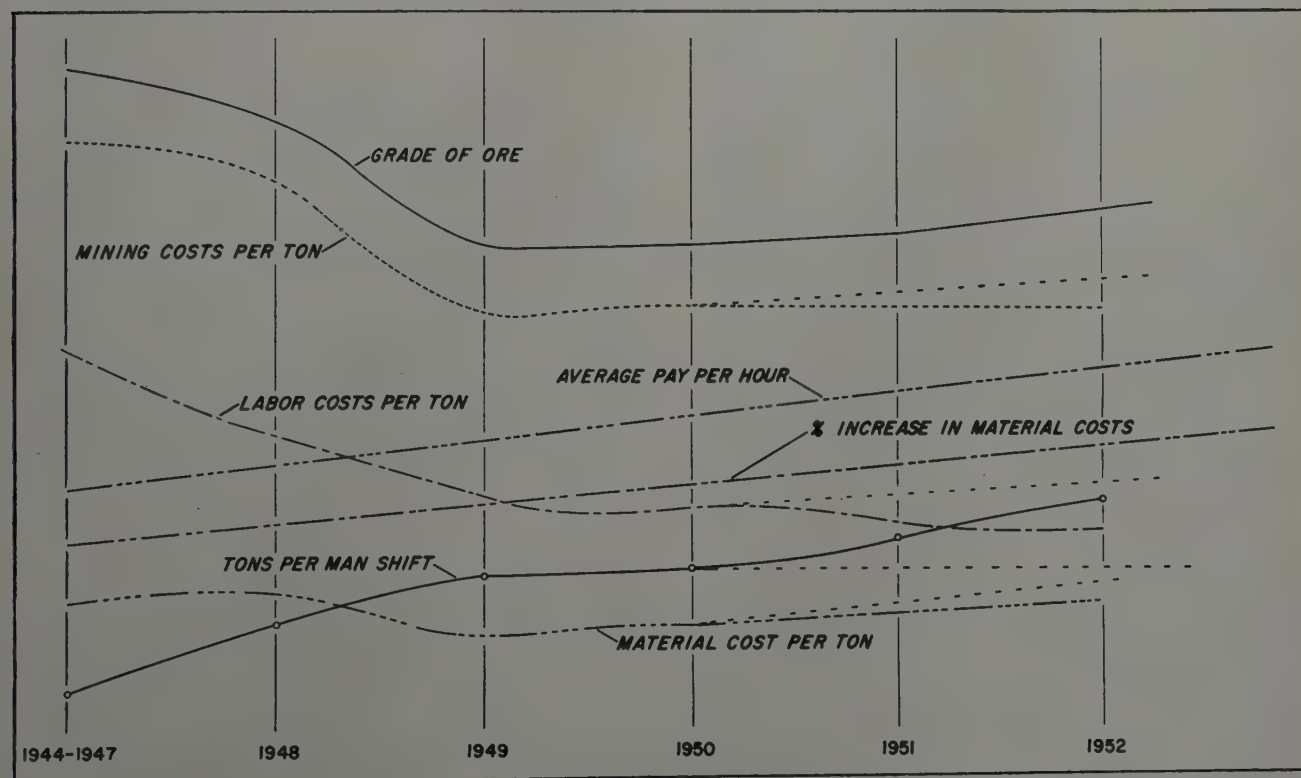


Fig. 2—Total mining costs during the 5-year period 1948 to 1952. The years 1944 to 1947 were used as a base period.

The year 1947 was chosen as a starting point because it was then obvious to operators everywhere that to cope with the high operating costs of low-grade nonferrous metal mines it was going to be necessary to improve mining methods. A general decontrol, the ending of the Premium Price Plan and the thawing-out of prices, and the close of the post-war boom occurred at this time. Mine operators were left with a low lead price and high costs for labor and materials. Fig. 2 shows the improvements made between 1947 and 1949 under the Tri-State car and can haulage system.

Necessary Mechanization

In 1949 another operating crisis was experienced. Cost and efficiency improvements had reached their limits, and it was necessary to head off the surge toward increased costs. This is demonstrated in Fig. 2 by the constant rise in labor and material costs and the flattening of the efficiency curves.

Many solutions to the problem were considered, but it was obvious after the study that the hauling and hoisting complications were the major items of the mining cost figure. Other costs could probably be held if these were reduced. After a consideration of ore reserves and the economics of each plan, it was decided to connect No. 1 and No. 5 mines by a 950-ft tunnel driven in such a way that it could be utilized for diesel truck haulage from the No. 1 mine to No. 5 mine shaft. Fig. 3 shows the tunnel connecting these mines and the haulage roads and old railroads. This system was chosen because the conventional two-ton balanced-skip hoisting-shaft, available for use at the No. 5 mine, eliminated the use of one can shaft of low capacity. The 18x14-ft tunnel was driven with a 104 Eimco diesel-mounted loader and three Koehring dump trucks, at an average daily advance of 11 ft. The tunnel was ventilated with a rubber-impregnated sail pipe. No churn drill hole could be driven from the surface because the tunnel

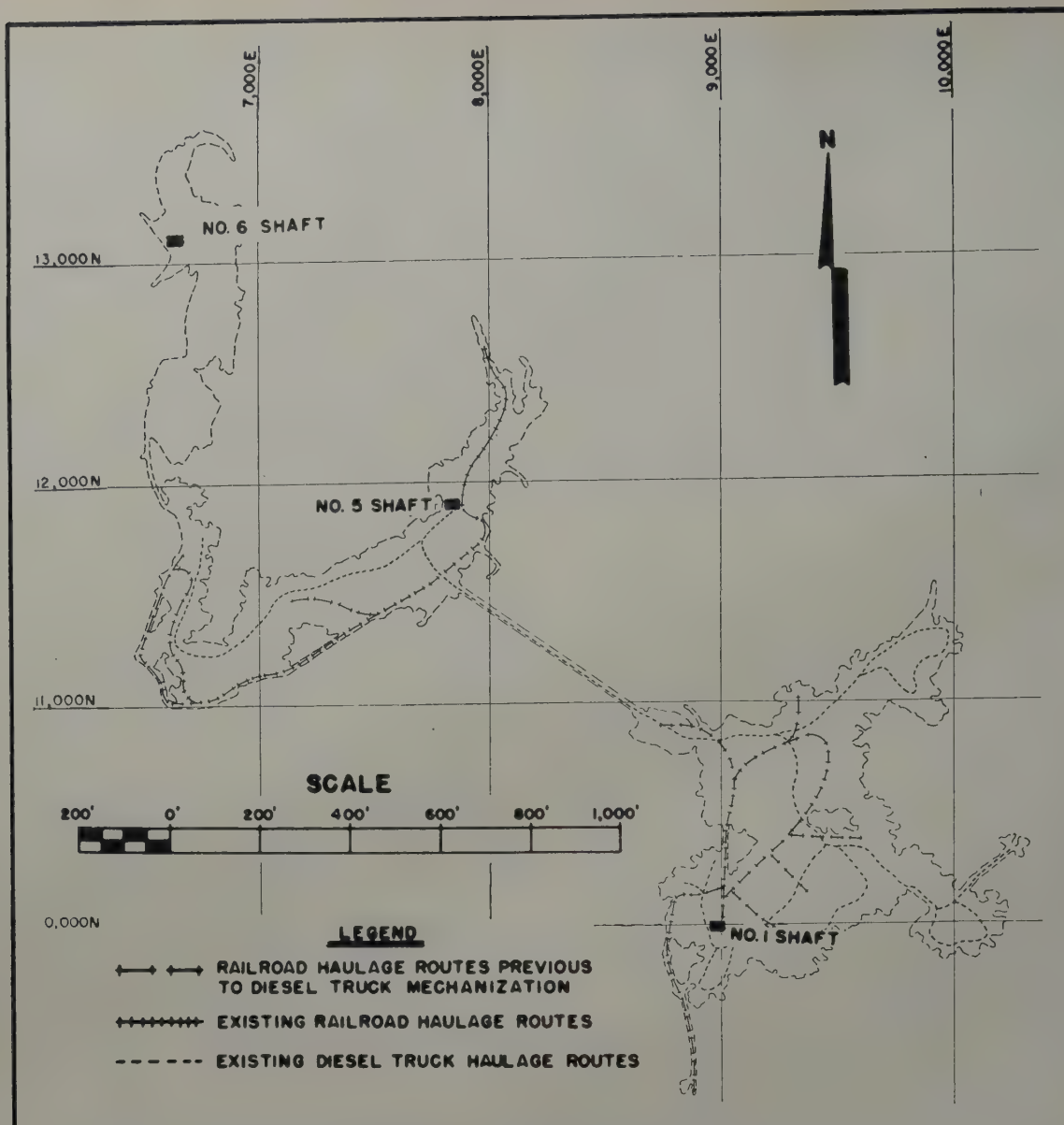


Fig. 3—Underground haulage routes, No. 1 and No. 5 mines. A 920-ft tunnel connects haulage routes and old railroads.

was under the reservoir which was used for storing mill tailings.

The drift work was carried on from the southeast end or No. 1 mine side on a contract basis. To speed the completion of the job, work was done simultaneously at the northwest end, or No. 5 mine side, on a daily pay basis. The ancient haulageways and mining and the conflict in production did not permit easy movement of equipment for disposal of waste rock from the drift on the No. 5 mine side. A unique system was devised to overcome this bottleneck, utilizing a D-4 bulldozer equipped with a three-drum slusher hoist mounted on the back opposite the blade. The broken rock from the night before was pulled back with the scraper, and the jumbo used for drilling the drift rounds was driven over the broken muck pile. In this way it was possible to begin drilling sooner after the shift started than it would have been with ordinary methods. The D-4 bulldozer then pushed the muck out of the portal into the adjacent old stope. This system was used on the northwest end for a distance of only 200 ft, which was the limit economically for pushing the

muck with the bulldozer, but it speeded the job to completion. Fig. 4 shows the high-lift type of machine, an Allis-Chalmers HD-9, used for loading, road building, and bulldozing.

The connecting tunnel, as well as roads around No. 1 mine, was completed and surfaced with drift rock by means of the bulldozer and a diesel tandem-drive road grader. The road grader, see Fig. 5, is still used for maintaining roads and the bulldozer for constructing new roads.

The 24x15-ft grizzly, finished at the time the tunnel was completed, received all the No. 1 mine ore, which was hauled through the lighted tunnel by Koehring 7-ton trucks. Fig. 6 shows the grizzly and Fig. 7 the dumptor truck discharging on screen.

The practice of using crawler-mounted jumbos was initiated in 1950, and started the trend toward improved breaking efficiency. Prior to this time drilling was done with a conventional post-mounted air-fed 2¼ or 2½-in. jackhammer devised years ago by St. Joseph Lead Co. The drilling problem as yet has not been solved, but there are constant improvements in jumbo results and jacking experiments.



Fig. 4—A high-lift machine used for road building and bulldozing also serves as emergency loader.

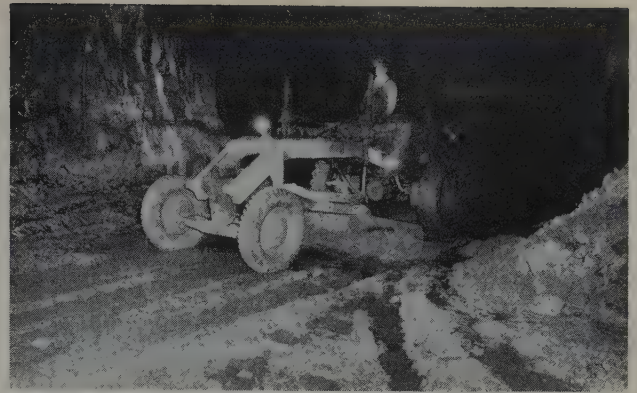


Fig. 5—This diesel tandem-drive road grader is used for maintaining roads and a bulldozer for building new ones.

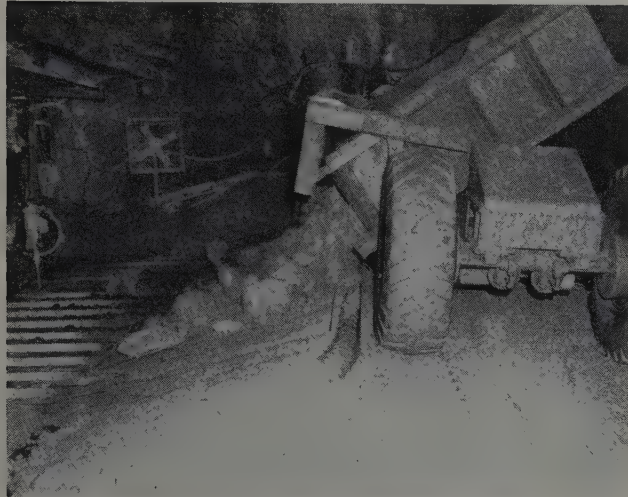


Fig. 6—The 24x15-ft grizzly was finished at the time the tunnel was completed and received all No. 1 mine ore.



Fig. 7—Dumpton truck discharging on screen. The Koehring dumpton has a 7-ton capacity and a loading height of 8½ ft.



Fig. 8—A crawler-mounted slusher hoist improved efficiency nearly 100 pct.



Fig. 9—Scraper loading into haulage truck, which has a loading height of 5½ ft. Note low headroom.

After the changes were made the No. 1 mine, producing 68 pct of the total plant output, showed improved efficiency, as indicated in Fig. 2 by the break in the lines after the year 1950.

No. 1 mine hoisting cost was greatly reduced by the utilization of the centralized hoisting system at No. 5 shaft. Manpower requirements were also reduced in the loading and haulage segments of the mining operation. The 104 Eimco, used to drive the

connecting drift, was employed on production where headroom and floor conditions were applicable. Tons per hour handled with this machine were as high as 100, but mud and irregular floor conditions resulted in high maintenance. It was then ear-marked specifically for development work, and only in emergency for production. Loading of the major part of the production was continued with the crawler-mounted slusher hoists and scrapers, see Figs. 8 and 9. Effi-

ciency with the slusher loader was improved almost 100 pct. Slushing into cans on railroad track limited the operation to the railroad and to use of small ore cans. Dragging time from face to rail-loading point was long in most cases because of the difficulty in building good railroad grade to within 50 ft of the face, especially over faults and breaks. Now, with drag loaders able to handle broken ore immediately in front of the machine, cable cost has been reduced 35 pct.

Haulage cost was lowered 50 pct immediately because the Koehring dumptor truck could negotiate grades of 8 to 12 pct over faulted areas. With the railroad system, an abrupt change in elevation necessitated moving hundreds of feet of track and building a fill to keep the grade under 3 pct.

Maintenance cost for locomotives, cans, and rail trucks on which the cans were hauled had been high, since track and cleanup crews constantly had to repair track and clean mud from moving parts of the equipment. Now the road grader keeps roads smooth and clean of rocks and mud. The Allis-Chalmers HD-9 makes fills or cuts very efficiently and is used with bulldozer attachment for dozing work. In Fig. 4 it is shown in emergency use as a loader, but this is not recommended, since track repair cost is too high.

Because the mechanized system had proved flexible and efficient, it was decided in 1951 to mechanize the No. 5 mine, developing from No. 1 mine the ore reserves being mined from No. 2 shaft, and closing No. 2 mine down.

A Low Headroom, High-Capacity Haulage Unit

Several problems were encountered in mechanizing the No. 5 mine. There was a low headroom of 8 ft, a long distance of 2300 ft around the porphyry contact, and grades to be negotiated. (The railroad system in use at this mine is shown in Fig. 3.) The orebodies require relatively close pillar spacing in some places where faulting and fractures have made it necessary, and therefore space for turning any equipment chosen was also a serious problem. This meant that a haulage unit must be devised with a speed equal to that of the dumptor, capable of carrying a load which would compensate for the distance traveled. The unit must have a small turning radius, a maximum loading height of 5½ ft, and a maximum headroom of 8 ft when occupied by the driver. It must be constructed to discharge on the large grizzly over the skip pocket at the shaft, as shown in Fig. 6.

The equipment chosen was a Caterpillar DW-10 rubber-tired tractor, diesel-powered, with hydraulic steering, pulling a newly designed, heavily constructed semi-trailer. This equipment was designed jointly by National Lead Co. personnel and Miles Landis of the Landis Steel Construction Co. of Picher, Okla. The trailer, which handles 14 to 17½ tons live load, was designed with hydraulically controlled side dump. Loading height is 5½ ft. The goose neck coupling between trailer and tractor was made especially heavy to reduce height and improve strength. This haulage machine, shown in Fig. 10, met the demands of the various problems.

Like the Koehring dumptor, which has a capacity of 7 tons and a loading height of 8½ ft, this unit has proved extremely rugged and flexible. Each unit has a speed of 18 mph. The average haul from any room or stope in the mine is more than ½ mile. The time required for a DW-10 tractor-trailer or a Koehring unit to make a complete trip is about 20

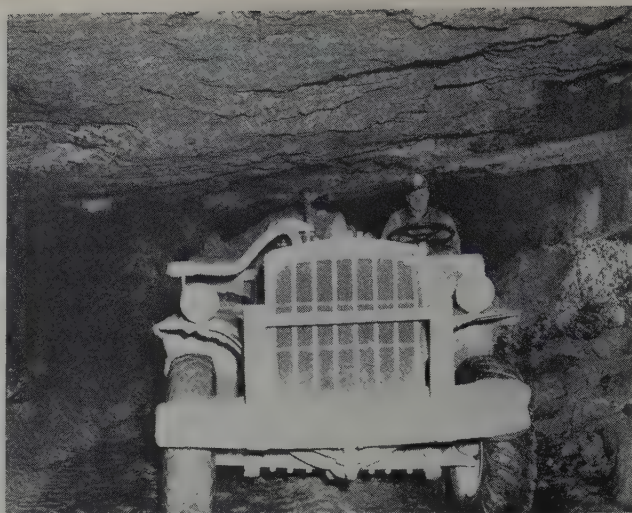


Fig. 10—Haulage machine designed to meet the problems encountered in National Lead Co. mines.

min. This haulage time, of course, is dependent upon the particular conditions under which the loaders are working, namely, ore fragmentation, size of muck pile, accessibility with trucks, type of loader, and headroom and pillar spacing. It is not uncommon for three trailers operating under the conditions described here to deliver to the skip pocket 600 tons of ore in an 8-hr shift, the Koehring trucks supplementing the production where conditions require. It must be remembered that here, as in all mining operations, only 5 to 6 actual hours of work are realized in an 8-hr shift. Lunch hours, transportation time to and from working places, and time for greasing equipment must be taken into account.

Attention is called to the adaptability of the two types of trucks, with emphasis on the efficiency of the smaller Koehring truck for development work. The short turning radius of 19 ft 6 in., together with the Koehring-designed driver position with respect to the load, are particularly advantageous for mine development. Experienced driver control in spotting these units around closely spaced pillars and under loaders is almost uncanny.

The DW-10 trailer unit, which is 35 ft long and 7 ft wide, requires more room for maneuvering, but here again, drivers can back or jack-knife the long trailer into a very small space. This is possible because of the 26-ft turning radius of the tractor and the hydraulic steering. Naturally it will become increasingly desirable to use this larger unit as haulage distances become longer, since it operates at lower cost per ton than does the Koehring truck.

The necessary variety of mine loaders was completed early in 1952 with purchase of the 25 TL Thew-Lorain turntable shovel, which has a 1½-yd capacity front-end loading boom. The Lorain shovel is applicable where headroom is sufficient, i.e., 9 ft 6 in. or more, and where the capacity can be utilized so that it does not affect the grade of ore too greatly. (This last condition is essential because the complex minerals must be fed to the concentrator in a constant grade of feed.) Drag loaders or Allis-Chalmers hydraulic high-lift loaders are utilized in working areas with lower headroom. The 104 Eimco is specifically assigned, as mentioned earlier, to development work in cuts, drifts, crosscuts, and narrow stopes.

Supervision of mechanized operations is extremely important, and foremen are equipped with jeeps

for greater efficiency. One foreman supervises loading, hauling, and hoisting, and another supervises drilling and breaking. This kind of leadership is not feasible in railroad mining operations.

The Underground Maintenance Shop

The maintenance department has been located at a central point for efficient performance. Here, as in many other mines throughout the country, all fabrication, welding, drill steel preparation, and rock drill, truck, loader, and jumbo repair are carried out in the shop. Maintenance rather than repair is emphasized because the sand-impregnated mud already mentioned finds its way into all moving parts of the mechanized equipment. It has been found that regular greasing keeps the mud out of bearings and moving parts, so that maintenance cost is reasonably low. Inspection reports are maintained, showing precisely the quantity and type of work performed on each piece of equipment and the number of hours of operation since the same work has previously been performed. Constant checks are made on motor efficiency and fuel and oil consumption, and exhaust fumes are kept within Bureau of Mines regulations at all times.

The ventilation system devised has eliminated the necessity for an exhaust treatment system on the production equipment. Fresh air forced down inactive shafts and some drill holes is picked up by underground circulating airplane-type fans, discharging at the hoisting shaft at No. 5 mine. An updraft condition exists at all times.

Spare haulage and loading units are kept in supply so that complete overhaul and maintenance practices can be followed. Mounted spare tires and other complete unit replacement parts are ready for quick changes when problems occur. The excellent tire wear and low tire replacement cost should be mentioned at this point. The 18x25 tire withstands the severe punishment of underground haulage. Re-

placement cost is low and in most instances the carcasses can be recapped with satisfactory results.

In summary it can be stated that if the mechanized system had not been adopted, the mining operations described here would probably not have survived the low lead prices between 1949 and 1952. It can be seen by a study of Fig. 2 that if costs had increased steadily, the cost curve would have intersected the grade of ore curve. Since grade of ore could not be increased, continued operation would not have been economically feasible.

Although actual mining cost per ton was not reduced in proportion to the raised efficiency factor, improvements made in ore handling, coupled with other advantages of mechanized equipment applicable to the complex orebodies of this property, made it possible to hold mining costs to a horizontal trend despite rising material and labor costs.

Prior to 1950 it was necessary to have 128 men in the mining operation, including those necessary for underground maintenance, hoisting, and grizzly. By the close of 1952 this crew had been reduced to 80 men. This reduction accounts for the upward trend of efficiency curves, Fig. 2, showing tons per manshift underground. Obviously the ideal condition would be to pay the highest possible wage and keep the mining cost curve as low as possible. It is possible to do so when the efficiency curve has this appearance.

Since the beginning of 1949, 1250 to 1400 tons of ore per day have been produced at this property. Increase in efficiency has enhanced the life of the property as well as the ore reserves. The difference between 1400 tons produced by 128 men and 1400 tons produced by 80 men is an efficiency improvement of 57½ pct.

Acknowledgments

The author expresses appreciation to Stuart Landrum, H. A. Neustaedter, T. J. Parkin, H. A. Powers, and J. A. Rood for their help in preparing this paper.

Technical Note

The Application of Geophysical Exploration in Civil Engineering

by George F. Sowers

THE geophysical methods of exploration employed so extensively in mining and petroleum investigations are now being used in civil engineering. Requirements differ from those of mineral ex-

ploration and therefore modified techniques have been developed.

The civil engineer is primarily interested in the uppermost soil and rock strata. In highway engineering the materials affecting design and construction may be found within 15 or 20 ft of ground surface, and data on soils and rocks is seldom needed for more than a depth of 100 ft even for the heaviest structures.

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TN 173L. Manuscript, Feb. 17, 1953.

Table I. Relative Merits of Geophysical Methods and Comparable Boring Methods

Method	Cost of Equipment, \$	Minimum Crew	Cost Per Lin. Ft Depth, \$*	Advantages	Limitations
Hand auger	10	1	0.50 to 1.00	Soil samples	Limited by gravel, ground water table; 30 ft maximum depth.
Powered hand auger	500	2	0.25 to 0.50	Soil samples	Limited to 6 to 8 ft depth.
Truck-mounted power auger	3000	1	0.30 to 1.00	Soil samples	Limited by boulders and ground water.
Wash boring	500	2	0.30 to 1.00	Washed samples	Cannot bore gravel.
Seismic	4000	3 to 4	0.25 to 0.50	Not hampered by boulders	Confirmation by boring necessary. No good if rock surface is irregular. Soil character only estimated. Requires explosives.
Resistivity	500	3 to 4	0.25 to 0.50	Not hampered by boulders	Confirmation by boring necessary. Soil characteristics estimated. Affected by strong electric current.

* Commercial cost including overhead and reporting of data.

The data required are depth and thickness of soil strata; composition and engineering properties of the soils, such as strength, permeability, and compressibility; depth to the rock and soundness of rock; and depth to ground water. In preliminary design stages data need be only approximate, whereas for the final design accurate detailed statistics are necessary.

Three methods have been found useful in civil engineering investigations: the variable-spacing-electrode resistivity, the constant-electrode-spacing resistivity, and the refraction-seismic. Others such as the various gravity measurements and the reflection-seismic have found only limited use in special situations.

The variable-spacing-resistivity method has frequently been employed in determining depth to bed rock and depth to ground water. It has been particularly useful in preliminary surveys of highway routes, when it is necessary to estimate the amounts of rock excavation over extensive areas. Occasionally the method has been used to determine depth and thickness of different soil strata, but only when the soils are relatively horizontal and uniform and when the resistances of the strata are widely different.

The uniform spacing or traverse method involves determining resistivity for constant depth over a wide area. Contour maps of high and low resistivity are then prepared which define areas having shallow or great depth to rock. Such maps have also been used to locate sources of sand and gravel, since these materials, when dry, have relatively high resistances.

The refraction-seismic method has been widely employed to determine depth to bed rock, especially where that depth has been 50 ft or more. It has been used with great success in exploring possible dam sites in areas where deep deposits of sands overlie the rock. The method will differentiate between dry gravel and rock, as the resistivity method may not do, and is especially advantageous for soils containing boulders, which can hinder drilling operations seriously. It has also been used to determine the boundaries of soil strata; in some cases it has been possible to identify the soils by their characteristic velocities.

Unfortunately the civil engineers who must decide which kind of exploration to employ have sometimes been oversold on the advantages of geophysical methods, which like any other indirect method have inherent weaknesses. They have not understood the limitations of the data obtained, and as a consequence engineers have been disappointed or dissatisfied with the results. To prevent misunderstandings, geophysicists who work with civil engineers should forewarn them.

In the first place, geophysical exploration provides the engineer with only part of the underground data he needs for design. It can indicate depth to rock and to ground water and in some instances can develop the stratification of soils, but it cannot identify materials accurately or indicate their physical properties. These data must be obtained by other methods.

In the second place, geophysical exploration sometimes provides inaccurate or misleading information when underground conditions are erratic or otherwise unsuited to the work. Unfortunately it is not always possible to determine unsuitability ahead of time. On one job in North Carolina seismic exploration indicated sound rock at a depth of 30 ft but subsequent exploration disclosed it at 15 ft, an error of 100 pct. On a site in Georgia the average error in determining depth to bed rock by electrical resistivity was 160 pct of actual depth and the minimum error 40 pct. In both instances the work was done by experienced crews under the supervision of geophysicists. Before any engineering work based on geophysical exploration is undertaken, borings should be made to confirm the data, and the entire project should be reviewed by competent geophysicists, geologists, and engineers.

In some situations characteristics of the site may prevent any geophysical work. Seismic work is ordinarily impossible in cities because of regulations regarding the use of explosives. On a few sites, electrical current in the soil renders accurate resistivity measurements impossible. For example, measurements made in an old fill to determine depth to virgin soil proved so erratic as to be worthless. The cause was deduced to be electrical currents generated by acid cinders in contact with junk metal that comprised what had been an old rubbish dump.

Information obtained by geophysical exploration is in many ways comparable to that obtained by wash boring and auger boring. The decision to use geophysical methods or direct methods depends largely on convenience and cost. Advantages of various methods are compared in Table I. In general geophysical equipment is expensive and relatively complex. Trained personnel and experienced supervision are needed to obtain satisfactory results. On the other hand, an experienced crew is able to secure large amounts of data in a short time.

Auger and wash borings utilize relatively simple, rugged equipment which is available on almost any fair-sized construction job. Even non-professional men can usually obtain good results. Shallow work can be done quickly, but when depths greater than 100 ft are involved and when coarse gravel and boulders are encountered, progress may become slow, if not impossible.

Laboratory Procedures For Determining Pelletizing Characteristics Of Iron Ore Concentrates

by T. E. Ban and L. J. Erck

A discussion of laboratory procedures used to determine pellet quality and to simulate handling and firing conditions. Strength-temperature relationships in pelletizing; effect of chemical additives on agglomeration; and crushing, abrasion, and impact resistance of the pelletized products are discussed with respect to five different iron ore concentrates.

IN most routine experimental work on products and processes, methods of operation and product evaluation are established from accurately controlled laboratory tests. With particular emphasis on the pelletization of fine iron ore concentrates, standardized procedures of laboratory production and product testing have been established at Cleveland-Cliffs Iron Co. Research Laboratory at Ishpeming, Mich. A program of this nature was used for investigating the agglomeration characteristics of various iron ore concentrates. The factors which control the process were studied and subsequently controlled for these investigations.

Pelletizing as applied to iron ore agglomeration involves the preparation of spheroidal masses from a fine moist concentrate, followed by high-temperature firing to produce a final product. The preparation of moist spheroids is generally designated as *balling*. This produces the so-called green pellets or balls. After these have been fired they are called *pellets*. Balling is accomplished by rolling moist concentrates in a revolving drum until small spheroids form by a snow-balling action. These balls are then heated in a shaft furnace, or a modification of this device, to a temperature high enough to cause agglomeration within the individual balls while they descend countercurrent to hot oxidizing gases. By this process the low strength balls are converted through various chemical and physical-chemical processes into hard pellets. With correct balling and firing, the pellets are strong, resist abrasive action and handling treatments, and provide a suitable blast furnace and open hearth feed.

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Discussion on this paper, TP 3552B, may be sent (2 copies) to AIME before Oct. 31, 1953. Manuscript Oct. 24, 1952. Chicago Meeting, September 1952.

In 1950 Tigerschiold and Ilmoni developed a working hypothesis to explain the formation and stability of green pellets.¹ Their work was presented on the basis of geologists' discoveries concerning the soil-water system and C. V. Firth's early postulates of green pellet formation.² Fundamental reactions occurring within fired pellets have been described by Cooke and Ban, who showed the microstructure changes caused by different firing temperatures.³ All these investigators have provided scientific clarifications of agglomeration by the pelletizing process and have opened avenues for greater development.

Mineral grain growth and ceramic bond formation are the two basic phenomena imparting coherency to particles within a fired pellet of iron ore. Each of these is established by the transfer of thermal energy to the mineral particles. Grain growth mechanisms are chiefly physical reactions wherein smaller particles consolidate into larger ones, with resulting decrease in surface energy conforming with the second law of thermodynamics. Ceramic bond formation is a chemical reaction usually between basic and acid metallic oxides of the gangue and ore minerals. CaO , MgO , Al_2O_3 , and FeO of Fe_3O_4 react in varying proportions with SiO_2 or Fe_2O_3 to form a great series of compounds or amorphous solid solutions. These product bond iron mineral particles together by a bridging action or by direct reaction with the surfaces of the particles. The result of the grain growth-ceramic bond formations is a consolidated, coherent compact of gangue and iron minerals formerly held together by much weaker bonds of interstitial water and forces common to dry, unconsolidated particles.⁴ With respect to iron ore agglomeration, these high temperature bonding reactions have been described by Schwartz, who referred to the bonds in sinters,⁵ and by Cooke and Ban, who referred to the bonds in fired pellets.⁶

The extent of the bonding phenomena and the consequent strength of fired pellets is governed by

some of the following firing conditions and properties of the unfired pellet: 1—mineralogical composition of the pellet, 2—particle packing within the pellet, 3—particle size range of the minerals comprising the green pellet, 4—maximum temperature to which the minerals are exposed and the retention period at this temperature, 5—the rate of attainment of and rate of recession from the maximum temperature, 6—the gas composition of the atmosphere in which the pellet particles are fired, and 7—the high temperature chemistry of the mineral particles and the additives of the pellets.

The mineralogical composition of the concentrate denotes the quality and quantity of thermally active compounds. Acid and basic oxides when present in critical amounts will materially promote interparticle consolidation by a ceramic bond formation, and iron minerals have grain growth properties peculiar to the ore. If the gangue constituents are present in abnormal quantities, pronounced slagging will occur within the fired pellet, and iron particles will be weakly bonded by excessive glassy constituents which inhibit the iron particles from interlocking by grain growth.⁶

Thermal barriers must be overcome before many ceramic reactions occur. The free energy of a chemical reaction must be of a negative value before the reaction occurs, and the greater the negative value the greater the reaction tendency becomes. At constant entropies and enthalpies, a negative free energy of reaction becomes more prominent with increasing temperature. A higher temperature generally increases a chemical reaction tendency and rate. This same higher temperature generally increases particle mobility and solid diffusive characteristics contributing to the grain growth mechanism.

A rapid rate of temperature increase lessens the compaction of a green pellet, owing to the rapid expulsion of air or bound water. This causes mineral particles to become less intimate for the bonding reactions. A rapid temperature increase also causes a duplex structure to result within a green pellet of magnetite. If sufficient heat is transferred to the center of a pellet prior to oxygen diffusion and magnetite oxidation, the core will consolidate to coarse-grained magnetite, while the shell or outside surface oxidizes to hematite. The coarse-grained magnetite core becomes less susceptible to subsequent oxidation by virtue of the coarse structure and the presence of slag coatings caused by the reaction of FeO with SiO_2 . The duplex-structured pellets, hematite shell somewhat separated from a magnetite core, have a lower strength index than pellets containing an homogeneous network of iron oxide. Rapid cooling of heated iron ore pellets causes bond and lattice strains from differential thermal contraction. This is common to quenched vitreous materials.

If the atmosphere of the fired iron ore particles is not of homogeneous composition, differential agglomeration will come about within the pellet because of particle chemistry differences. Hematite, magnetite, and the lower oxides of iron definitely possess different properties of agglomeration. If some of the iron minerals are contiguous to reducing substances during firing, the agglomeration characteristics will differ from particles fired in an ambient atmosphere that is oxidizing in nature.

In a generic sense, ideal agglomeration is afforded by the following conditions: 1—the presence of an optimum quantity of slag-forming constituents within the concentrate, 2—the use of an optimum

firing temperature, 3—a slow rate of temperature increase during firing, 4—a slow rate of temperature decrease after firing, 5—a long retention time at the optimum firing temperature, 6—a high degree of compaction, and 7—an homogeneous oxidizing ambient atmosphere during the firing period.

The Cleveland-Cliffs Research Laboratory staff has developed an apparatus for evaluating the relative response of various fine concentrates toward pelletization and for enabling a study of the various additives or factors which aid the process. This apparatus was designed and operated to simulate the conditions imposed by the shaft furnace firing of iron ore pellets. For this purpose, some of the operating conditions of a pilot plant shaft furnace were obtained and incorporated within the laboratory device.

Shaft furnace firing of green iron ore pellets is carried out by the placing of wet pellets on the top of a shaft hearth containing pellets in the process of being fired. As the charge descends into the shaft by gravity, the wet pellets are fired by a countercurrent stream of combustion and heat transfer gases. Varying temperature zones within the furnace perform the following consecutive operations on the descending wet pellets: 1—drying, 2—preheating, 3—firing to a maximum temperature, and 4—cooling.⁷

The regions of the hot zones depend upon the temperature of the hot gases, the chemistry of the pellets, and the rate of descent and ascent of the pellets and gases respectively. Ideal shaft furnace operation requires isothermal temperature zones transverse to a path of uniform gas flow and charge descent. It is very difficult to obtain these conditions because of pellet particle segregation and inter-pellet consolidation, both of which disturb ideal gas flow and temperature patterns.

The regions of different temperature zones within the furnace shaft can be varied by operational controls, but a specific zonal region can also be a function of the high temperature chemistry of the ore pellets. If pellets contain an intrinsic fuel, magnetite or an additive which can exothermally oxidize with the gases, a hot firing zone can literally be floated within the shaft when relatively low temperature gases are allowed to ignite the pellet burden and maintain continual combustion within the descending charge. In this manner descending pellets of magnetite can be ignited with oxidizing gases of 1000°C and a zone of burning pellets can be established that is 1300°C in temperature. The position and maintenance of this zone gives rise to a longitudinal temperature gradient within the furnace shaft, which consequently describes the changes of pellet temperature with time.

The locations and temperatures of various zones in a pilot plant shaft furnace were determined with probing and stationary thermocouples during many pelletizing tests. From these data and from the geometry of the furnace shaft, the rate of pellet temperature change, Fig. 1, was computed. The two curves of Fig. 1 show the results of these computations from furnace tests using pellets of hematite and magnetite. The hematite pellets contained no fuel and the maximum temperature was imparted entirely from the temperature of the hot gases from the combustion chamber. For this reason hematite pellets had a relatively low rate of temperature increase. Magnetite pellets, on the other hand, burned in a zone above the entrance of the hot gases and gave rise to a rapid temperature increase. The rate of temperature decrease was approximately the same,

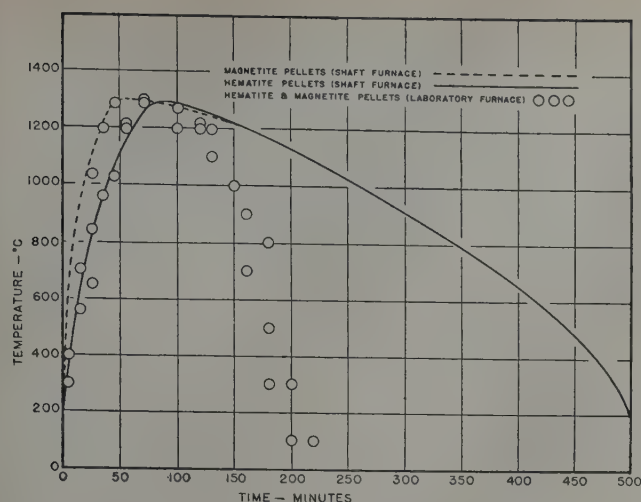


Fig. 1—Time-temperature relationships during pilot plant and laboratory firing of iron ore pellets.

however, regardless of the kind of ore comprising the green pellet.

Air and fuel oil combustion products were the principal gases used for the pellet heat transfer and firing medium respectively. The heat transfer gases were admitted to the pellets from the furnace bottom and ascended countercurrent to the shaft burden in a quantity which was adjusted to the specific heat capacities of ore and air. These are 0.262 to 0.268; usually, therefore, 1 lb of air was used to abstract the heat from 1 lb of ore pellets. The combustion gases were transversely admitted to the pellets in the upper portion shaft in a quantity adjusted to the thermal requirements of the pellets. This quantity naturally varied from test to test, but averaged about 0.8 lb of combustion products to 1 lb of pellets. The composition of the combustion products in turn was subject to variation in that different quantities of excess air were used to provide different temperatures. Combustion products analyzed on the order of 6.42 pct CO_2 , 9.4 pct O_2 , 77.5 pct N_2 , and 6.7 pct H_2O . Both the heat transfer gas and combustion gas mixed near the pelletizing hot zone, so that an overall 1.8 lb of gas were admitted to 1 lb of ore; the gas analyzed about 3.0 pct CO_2 , 15.3 pct O_2 , 78.6 pct N_2 , and 3.1 pct H_2O .

Laboratory Procedure for Firing Pellets

Shaft furnace pellets undergo a temperature change with time during the transition from the wet state to the fired state in accordance with the curves of Fig. 1, and about 1.8 lb of gas containing 15.3 pct O_2 are admitted for each pound of pellet charge. This knowledge was applied to the operation of the laboratory apparatus, see Fig. 2, used to simulate the shaft furnace firing conditions. Apparatus consisted of an electrical, globar combustion tube furnace equipped with provisions for passing gas through the tube at controlled flow rates. Nichrome boats which served as the pellet-charging devices were admitted through the open end of the combustion tube, passed countercurrent to the gas stream, and removed through the upstream end of the tube. The boat charge consisted of five wet pellets, each of which was about 35 g in weight and 28 mm in diam. These were formed in a laboratory balling drum by use of procedures described by Tigerschild and Ilmoni.¹ The drum products were stored in sealed containers and retained for the subsequent firing tests. To min-

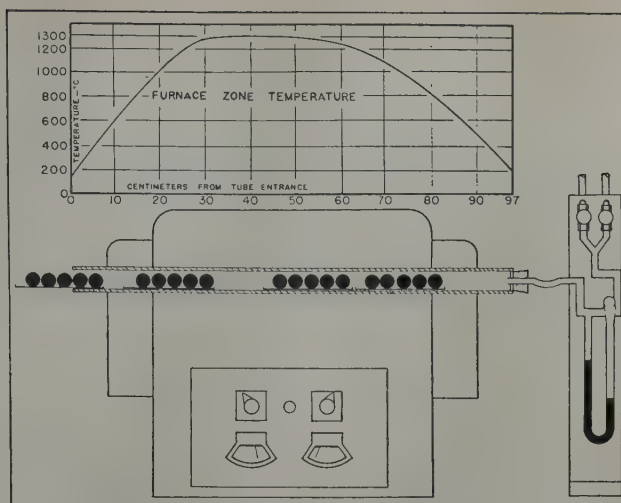


Fig. 2—Apparatus for firing iron ore pellets.

imize pre-drying effects, green pellets representing a specific balling drum product were taken from the containers and placed on the boat approximately 1 min before the actual firing operation.

During operation of the laboratory apparatus pellet-laden boats were admitted one by one into the combustion tube and moved periodically through the tube so that pellets could approach the consecutive temperature changes of shaft furnace pellets. For this purpose a standardized firing schedule was developed from a knowledge of the furnace temperature gradient curve of Fig. 2 and from the heating rate curves of Fig. 1. When fired by this schedule, individual pellets were periodically changed in temperature in accordance with the circular points of Fig. 1, while a countercurrent stream of air was passed over the pellets at a rate of 3.50 liters per min. This air flow corresponded to 1.23 mols of O_2 per mol of Fe_2O_3 , which in turn was roughly equivalent to the oxygen supply of iron ore pellets during the shaft furnace firing operation.

The graph of Fig. 1 denoted several instances where the laboratory firing schedule did not exactly duplicate the firing schedule of the shaft furnace. The laboratory cooling rate was more rapid, the heating rate was a mean of the pilot plant rates for hematite and magnetite, and the heating rate varied slightly from pellet to pellet.

These discrepancies were tolerable. It would have been impractical to allow laboratory pellets to cool for 10 hr, and the slight variations of the heating rates were of a small order when contrasted to the normal variations of heating rates within a shaft furnace. The results of the laboratory tests were to be relative rather than absolute. For this reason the shaft furnace firing schedule was approached but was not attained.

Pellet Quality Determinations

The quality of fired pellets can be determined by different materials tests to denote the toughness, hardness, and compressive strength. Each of these relates a measure of pellet particle coherency. Iron ore pellets should be able to withstand the crushing forces imposed by furnace and stockpile stacks and the impact and abrasive forces imposed by conventional iron ore handling and shipping. The physical properties that make it possible for pellets to endure these forces are generally related, although not def-

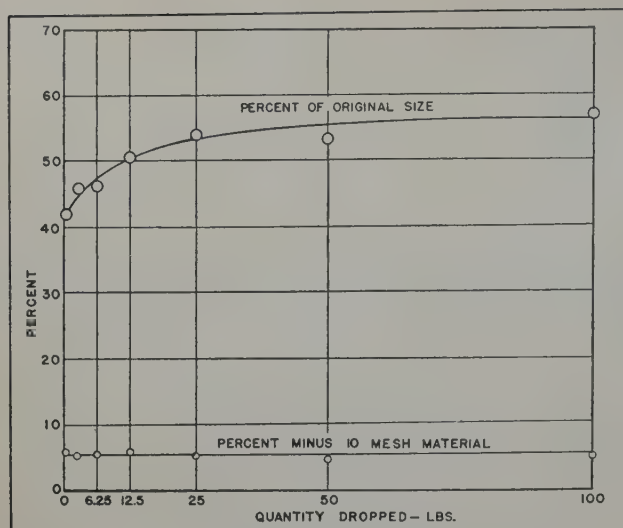


Fig. 3—Drop test relationship between pellet quantity and subdivision.

initely. Brittle pellets can conceivably withstand great compressing loads, yet will shatter remarkably from impacts. Tough pellets, likewise, can be comprised of coarse particles which readily abrade from the surfaces. For these reasons, each specific strength property was individually tested by the use of standardized materials test procedures. These were the crush, impact, and abrasion tests used to denote pellet strength, toughness, and hardness, respectively.

Strength is a measure of ability to endure stress. In the case of slow diametric compression of spherical material, stress is a function of sphere size and compressing force. When like-size spherical pellets are subjected to a slow diametric force, the strength and maximum unit stress are directly proportional to the crushing load.⁸ Consequently the pellet strength designated herein was approximated from a measurement of the maximum load imposed on an individual pellet, or the extent of the compressing load which caused the pellet to crush.

The tests were performed by compressing a pellet between a hydraulic piston and a fixed steel plate. The hydraulic pressure required to crush the specimens was indicated by a Bourdon gage. The pressure was converted to pounds of load, in turn recorded

as the pellet-crushing strength. Pellets representing a specific test invariably yielded strength values which were not identical. This was influenced by: 1—pellet size differences, 2—pellet shape differences, and 3—inadequate crushing techniques wherein the crushing force was not applied at a constant rate. Laboratory procedures only minimized the effect of these factors and the differences in crushing loads for pellets representing a specific batch were consequently minimized. Therefore, the average crushing load of 20 individually tested pellets was used to represent the pellet-crushing strength.

The abrasion test was developed to find a means of expressing the relative resistance to abrasive action offered by the surfaces of pellets. This abrasive action was caused by inter-pellet rubbing induced by pellet contact and movement in a small rotating drum. The drum consisted of a short cylinder mounted axially on a variable speed motor. The dimensions were 16 in. diam by 6 in. long. Exactly 1000 g of pellets, about 35 in number, were placed into the drum and the drum was rotated for 200 revolutions at 24 rpm. This rotational velocity was about 50 pct of the critical velocity so that the pellet movement was largely a cascading and rolling action. When the 200 revolutions were completed, the pellets and the fine dust produced from the test were removed and weighed separately. The weight of the abraded dust, which was all —65 mesh material, was used to express the results of the abrasion test. The intact abraded pellets and the fine dust produced from the surface were the only products of the test.

The impact test was developed for determining the toughness of the pellets or the relative resistance to subdivision from impact forces. These forces were imparted by dropping the pellets successively from predetermined height to a solid steel plate surface.

Exactly 1000 g of laboratory pellets, about 35 in number, were measured for size and placed into a container mounted 33 1/3 ft above a 1/2-in. steel plate surface. The container was fixed with a trap-door exit from which the pellets were allowed to drop through a 4-in. pipe guide onto the plate. The pellets and fragments were successively dropped three times and a screen analysis through 28 mesh was made of the final product.

A strength index, or the percent of the original particle size, was computed from the screen analysis

Table I. Determination of Average Particle Size by Screen Analysis by Coghill's Method

Original Product				Dropped Product			
Screen Size		Screen Analysis, Wt, Pct	Ordinal No.* (Arbitrary)	Wt, Pct	Screen Analysis, Wt, Pct	Ordinal No.† (Arbitrary)	Wt, Pct
Mesh	In.						
1 1/4	1.248		0	65.3	0.0	0	
1	1.050	65.3	1	69.4	8.0	1	8.0
3/4	0.742	34.7	2		22.8	2	45.6
1/2	0.525		3		27.7	3	83.1
3/8	0.371		4		13.8	4	55.2
3	0.263		5		10.6	5	53.0
4	0.185		6		3.6	6	19.8
6	0.131		7		3.3	7	23.1
8	0.093		8		1.4	8	11.2
10	0.065		9		1.4	9	12.6
14	0.046		10		1.0	10	10.0
20	0.033		11		0.7	11	7.7
28	0.023		12		0.6	12	7.2
-28	0.023		15		5.1	15	76.5
134.7	= 1.347	100.0		134.7	413.0 = 4.130		413.0
100					100		

* This ordinal number is between 1 and 3/4-in. screen size. By interpolation the average size equals: 1.050 - (0.347 (1.050 - 0.742)) = 0.943

† This ordinal number is between 3/8 in. and 3 mesh screen size. By interpolation, the average size equals: 0.371 - (0.130 (0.371 - 0.263)) = 0.357

The percent original size-strength index = $\frac{\text{Average size after impact}}{\text{Average size before impact}} = \frac{0.357}{0.943} = 37.9 \text{ pct}$

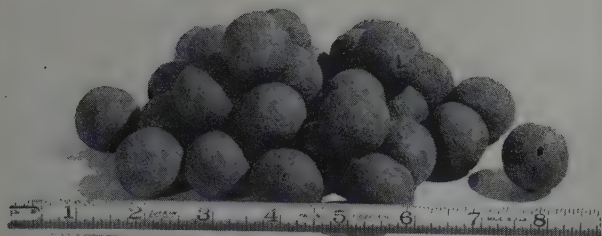


Fig. 4a—Intact pellets before testing. Pellets of hematite concentrate, 1000.0 g, containing 2½ pct limestone and fired at 1300°C in laboratory apparatus.

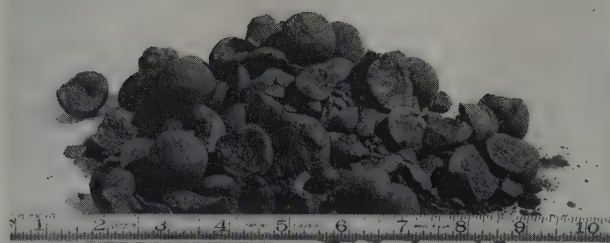


Fig. 4b—Broken pellets after drop testing, 33 1/3-ft drops onto a ½-in. steel plate surface. Drop test results: 3.5 pct —10 mesh material; 59.4 pct original size.

of the pellets before dropping and the screen analysis of the pellets and fragments after three drops. Coghill's method of mean mesh determination was adapted for this purpose.⁸ This provided a means of establishing the average size of the particles directly from the screen analysis. An example of this determination is shown in Table I.

This index, along with the percent of fine material (pct —10 mesh) related the toughness of the pellets, or the ability to endure the shock of impact forces. In effect this was provided from the ratio of the average particle diameter after testing to the average particle diameter before testing.

Significance of the Product Tests

The drop tests were carried out on 50-lb samples of pellets produced from the pilot plant shaft furnace and on 1000-g batches of pellets produced in the laboratory furnace. With the percent original size as a strength index, the severity of the test appeared to be a function of the quantity of pellets used for a specific test. When 50 lb of pellets were dropped one at a time, a very small number of pellets were exposed to the greatest impact; that is, only a few pellets from each 50-lb batch received the primary impact because of the relatively small projected area directly under the drop test guide. Pellets directly on the bottom of a load received the steel plate impact, while pellets in the center or top of the load were subject to hindered falling and a resulting cushioning due to the mass effect. This dropping condition did not exist for the 1000-g drop test because nearly every pellet and fragment was subjected to an impact force which was a function only of the particle mass.

Several tests were conducted to determine the mass effect of the drop test and to determine the reliability of results procured from a specific test. Approximately 700 lb of pellets of a uniform quality were thoroughly mixed for these tests and varying quantities were cut out from the entire sample. These quantities consisted of two 100-lb portions, two 50-lb, two 25-lb, two 12.5-lb, two 6.25-lb, and eight 1000-g portions. Each portion was dropped en masse with the exception of four 1000-g portions which were dropped in a manner to conform with single particle impact. This was accomplished by individually dropping each pellet and pellet fragment onto the cleaned plane surface area. This type of drop test naturally afforded the most drastic impact, or the most favorable conditions for particle subdivision by impact. The graph in Fig. 3 shows the drop test results from using different quantities of pellets for a specific test batch. It was observed that the percentage of —10 mesh material produced from the impacts was independent of the quantity

dropped, and the percent original size decreased with decreasing quantities of dropped material. This was understandable because the greater impact forces were imparted to a greater percentage of the pellet charges. The production of fines (pct —10 mesh material) was caused by both abrasion and impact, abrasion being more severe with the larger quantities. The fines produced from the impact of dropping smaller quantities of pellets were compensated for by the normal abrasion of dropping large quantities. The net result of both these effects yielded an independent value to the fines production with dropped pellet quantities.

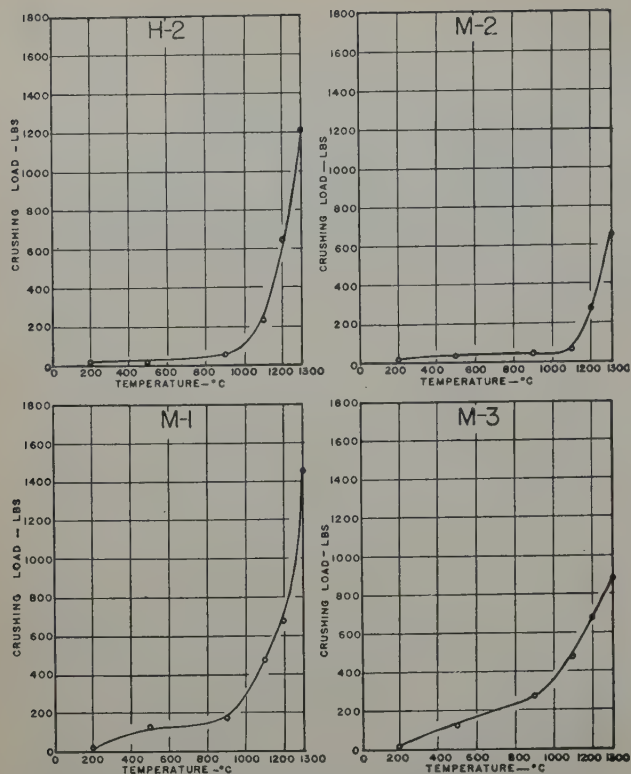
Four 1000-g batches of —1¼ +1 in. pellets were hand picked for uniformity of quality and were given an identical drop test to determine the reliability or degree of reproducibility of the test. The results as indicated by the fines production and percent original size did not vary from an average by more than 30 or 7 pct respectively. This discrepancy was tolerable and of small enough magnitude to warrant the drop test procedure as a reliable measurement of pellet toughness.

Figs. 4a and 4b show 1000-g batches of iron ore pellets before and after the three 33 1/3-ft drop tests. These pellets were composed of fine hematite concentrate containing 2½ pct limestone and were fired to 1300°C in the laboratory apparatus. The drop test results indicated that these pellets had a relatively high impact resistance for iron ore pellets, 3.5 pct —10 mesh and 59.4 pct original size. Some direct-mined ore specimens subjected to the same drop test, —1¼ +1 in. specimens, 1000 g in weight, 3 drops of 33 1/3 ft, had a greater impact resistance than the pellets. Below are listed some drop test results on several kinds of iron ore from the Cleveland-Cliffs Iron Co.:

Item	—10 Mesh, Pct	Original Size, Pct
Cliffs shaft lump ore	2.9	79.4
Tilden siliceous ore	1.7	70.9
Cliffs group ore	9.2	28.8

Some of these relatively high values may be greater than requisite values. For this reason the drop test results from the iron ore specimens were used for comparative purposes rather than as standards for iron ore pellets.

The most expedient method of measuring the degree of particle coherency within an iron ore pellet is the crushing test. A specific test carried out as a measurement of value, such as the crushing strength of fired pellets, is only as good as the degree of reproducibility of the test. If values are assigned as crushing strengths of pellets treated in a standard manner, the values should be attained with reason-



Figs. 5 to 8—Strength temperature relationships of iron ore pellets.

able accuracy by repetition of the test. A measure of this reproducibility is denoted by the probable deviation of the mean of a series of measurements, and the magnitude of this deviation indicates the precision of the measurements. This is mathematically determined by use of the equation:

$$Q = 0.674 \sqrt{\frac{1}{N(N-1)} \sum r^2}$$

wherein Q equals the probable deviation of the mean value, r the residual of the measured value and the

mean value, and N the number of measurements. In effect, this signifies that the probability is one to two that the mean value will have a residual denoted by the probable deviation.

Table II gives the individual crushing values of 15 kinds of iron ore pellets fired by the laboratory firing schedule to a maximum temperature of 1300°C. The significance applied to a crushing value for a series of pellets is indicated by the determination of the probable deviation of the mean of 20 individually crushed pellets. The figures indicate the average crushing values, the probable deviations of the average values, the best values, and the probable deviations of the best values stated as percentages. It was observed that the probable deviations of the best values were no larger than 12 pct and no less than 2.3 pct for the 15 sample measurements. These small deviations indicated that the laboratory firing and crushing tests were reasonably reproducible.

The abrasion resistance of pellets is dependent upon both the particle coherency and the friction factor of pellet surfaces. Coarser ore particles tend to increase the friction factor, and these coarser particles abrade more readily than equally bonded fine particles. This logic was substantiated from the comparison of abrasion test results of pellets comprised of the original ore concentrates and pellets comprised of reground ore concentrates, see Table III.

Results of Laboratory Pelletizing Tests

The laboratory tests were carried out on five different iron ore concentrates, designated as M-1, M-2, M-3, and H-1, H-2. The chemical and screen analyses of these are presented in Table IV. M-1 and M-3 represent two different concentrates of natural magnetite produced from magnetic concentration circuits. H-2 and M-2 represent concentrates of artificial hematite and artificial magnetite produced by roasting iron sulphides under different atmospheric conditions. H-1 represents a flotation concentrate of specular hematite. Each of these ores possessed peculiar properties, and as such responded differently toward agglomeration by pelletizing.

Table II. Crushing Strength of 20 Randomly Selected Pellets Expressed with Probable Error of Average Value

Additives, Lb Per Ton Conc. Condition	Pellets fired at 1300°C														
	H-1			H-2			M-1			M-2			M-3		
	10 b 50 ls	10 b 50 ls MR	10 b RG	10 b	10 b	10 b	10 b	10 b 40 c Surface	10 b 30 c	10 b	10 b	10 b 50 Pct RG	10 b 10 bx	10 b	10 b
1	1180	1460	730	670	1250	900	900	600	680	1250	600	650	900	630	790
2	1200	1020	620	630	1550	720	620	1000	580	3000	400	540	500	430	610
3	2500	3000	720	570	1250	750	570	1000	410	1750	600	490	450	830	870
4	1000	870	460	640	1250	550	800	900	570	940	300	620	1000	740	2500
5	1500	840	500	540	1250	480	1030	860	570	1250	300	680	400	520	610
6	3250	860	570	440	900	3000	1050	730	380	1250	550	600	1500	820	460
7	3750	1100	840	500	1250	560	950	500	160	1250	530	900	800	540	700
8	2250	2250	990	520	2750	800	2400	1000	500	1160	620	650	500	1000	670
9	880	1100	520	600	1050	840	640	910	530	1500	1100	440	1000	740	1100
10	1000	2860	650	430	670	760	950	550	600	1090	300	350	900	730	840
11	2500	1100	1070	540	980	4250	1060	540	720	1250	600	650	700	550	910
12	2000	1500	770	720	1110	910	890	600	700	3000	1100	620	800	550	680
13	1200	2500	820	620	620	630	820	1100	540	1250	300	530	600	420	880
14	2500	2300	1000	520	1050	800	1900	700	400	1250	680	1900	1000	790	840
15	2000	3840	900	580	1250	940	2000	900	330	1110	520	600	2500	720	660
16	2000	1020	1200	460	1250	660	950	1000	500	1500	3100	790	800	240	1060
17	1250	2250	850	630	940	410	1030	640	590	1500	400	730	1100	540	850
18	800	1100	840	760	2000	700	550	720	900	950	500	530	1200	800	820
19	1200	930	680	530	870	830	910	780	360	1500	2400	780	1000	930	860
20	1500	1100	600	550	1250	700	900	640	550	1250	400	730	900	540	930
Average — M	1773	1650	769	572	1224	1009	946	780	528	1450	765	689	927	653	882
Probable dev. — Q	±129	±138	±30	±13	±54	±99	±54	±30	±23	±68	±88	±32	±53	±29	±38
Best value	1800	1700	770	570	1220	1000	950	780	530	1450	760	690	930	650	880
Probable dev., pct	±7.2	±8.4	±3.9	±2.3	±4.4	±9.9	±5.7	±3.8	±4.3	±4.7	±11.6	±4.6	±5.7	±4.5	±4.3

The additives were added in pounds per ton of ore.
b denotes bentonite clay.
c denotes pulverized anthracite coal.
MR denotes a magnetically roasted conc.

ls denotes pulverized limestone.
bx denotes borax.
RG denotes a reground concentrate.

Table III. Summary of Pellets Test Results

Additives, Lb Per Ton	H-1				H-2				M-1			M-2		M-3
	10 b 50 ls	10 b 50 ls	10 b	10 b	10 b	10 b	10 b 40 c	10 b 10 c	10 b 30 c	10 b	10 b	10 b	10 b 10 bx	10 b
Conc. Condition	MR		RG		RG		Sur- face			RG		50 Pct RG		
Avg Pellet, Diam, In.	1.132	1.050	1.165	1.180	0.998	1.015	1.015	1.035	1.020	1.125	1.052	1.042	0.975	0.975
Screen Analysis of Pellets after 3-33-1/3' Drop Tests — Pct Weight Retained														
Mesh														
1	52.3	20.2	8.0	2.9	0.0	0.0	0.0	5.7	0.0	18.0	0.0	0.0	0.0	0.0
3/4	20.6	34.6	22.8	22.2	33.6	20.3	37.7	46.3	29.4	35.9	0.0	5.7	20.8	33.1
1/2	15.7	22.4	27.7	42.6	19.2	24.3	15.1	8.9	27.3	9.2	21.0	27.0	24.3	13.5
3/8	3.4	12.8	13.8	10.6	22.8	25.0	19.7	11.2	14.3	14.3	41.5	33.9	21.0	17.6
3	2.5	4.2	10.6	6.6	10.8	14.0	10.1	12.5	11.7	8.6	18.0	17.3	14.8	13.0
4	0.9	1.6	3.6	2.8	4.8	7.1	6.0	3.7	4.1	4.1	8.8	6.6	4.6	7.3
6	0.6	0.9	3.3	2.2	2.4	3.3	3.2	2.7	2.6	2.1	3.5	3.2	3.6	3.9
8	0.3	0.4	1.4	0.9	0.9	1.8	1.8	1.5	1.6	1.1	2.0	1.5	2.3	2.0
10	0.2	0.4	1.4	0.6	0.8	1.3	1.2	0.9	1.1	0.7	1.5	1.1	1.6	1.7
14	0.2	0.3	1.0	0.7	0.6	0.9	1.0	0.6	0.8	0.7	1.0	0.8	1.1	1.3
20	0.1	0.3	0.7	0.6	0.4	0.6	0.7	0.4	0.6	0.6	0.6	0.5	0.7	0.9
28	0.1	0.2	0.6	0.5	0.2	0.4	0.6	0.3	0.5	0.4	0.5	0.4	0.5	0.7
-28	3.1	1.7	5.1	6.8	3.4	1.0	2.9	5.3	6.0	4.3	1.6	2.0	4.7	5.0
Avg diam after drop, in.	0.675	0.555	0.355	0.325	0.381	0.359	0.386	0.390	0.336	0.441	0.283	0.321	0.313	0.316
Pct original size	59.4	52.8	30.5	27.5	38.2	35.3	36.2	34.8	33.0	39.2	27.4	30.7	32.1	32.4
Pct -10 mesh from drop	3.5	2.5	7.4	8.6	4.6	2.9	5.2	6.9	7.9	6.0	3.7	3.7	7.0	7.9
Crush test load, lb	1800	1600	750	550	1200	1000	780	950	550	1450	750	700	900	650
Abrasion test, pct -65 mesh	1.2	1.7	0.9	4.0	0.5	0.6	2.5	0.7	1.3	1.8	0.4	0.9	1.2	2.1
Pellets per 1000 g, no.	27	34	25	24	35	33	33	31	32	24	29	30	36	36
Pellets, wt, g	37.0	29.4	40.0	41.6	28.5	30.3	30.3	32.3	31.3	41.7	34.5	33.3	27.8	27.8
Pellets, volume, cc	12.5	9.9	13.5	14.1	8.4	8.9	8.9	9.5	9.2	12.2	10.1	9.7	8.1	8.1
Density of pellet gm/cc	2.97				3.36					3.42			3.44	
S.G. of ore gm/cc	4.23				4.86					4.44			4.40	
Porosity of pellet, pct	29.6				30.9					23.1			21.9	

Prior to balling, the ore concentrates were blended in the dry state with 1/2 pct by weight of bentonite and, in certain instances, with various quantities of other pulverized additives. These blends were further mixed with about 10 pct water and each mixture used as the laboratory balling drum feed. The balls were carefully formed by the method described by Tigerschild and Ilmoni,¹ to produce -1 1/8 +1 in. spherical compacts. These were fired from the wet stage to the fired stage in the manner outlined in the laboratory pellet firing procedure. The fired pellets were tested in the manner outlined in the pellet-testing procedures.

A swelling variety of bentonite was added as a pelletizing aid to all the iron ore concentrates. By

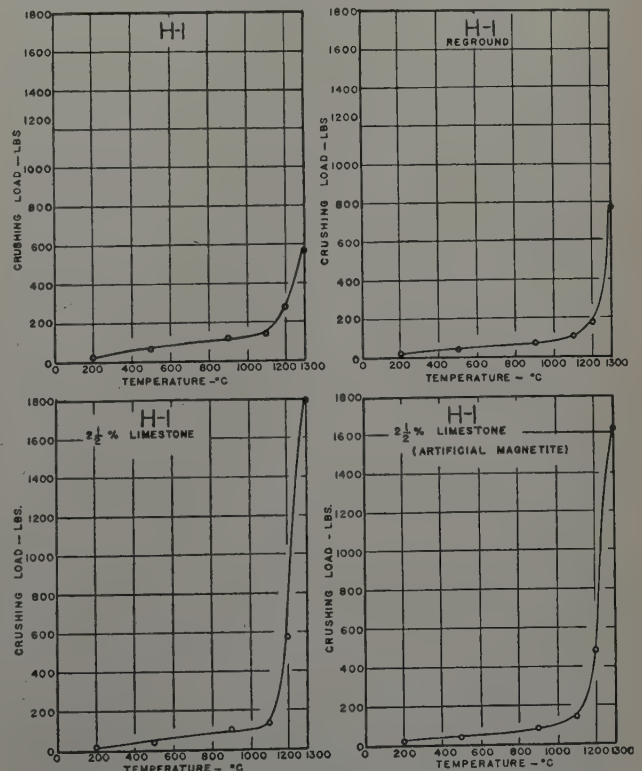
drying stage of firing, and the high alumina content of bentonite aids the ceramic bond formation.

The knowledge of these benefits, coupled with the low cost of the reagent, has made bentonite usage attractive to the pelletizing process. For these reasons, bentonite was added to all the ores used for the standardized tests. Ten pounds of bentonite were added to each ton of ore concentrate. To aid ceramic bond formation within the fired pellets, pulverized limestone and borax were added to ores difficult to pelletize. Limestone, after coincidental calcination during firing, provided active CaO which reacted at

Table IV. Chemical and Screen Analyses of Iron Ore Concentrates Used for Pelletizing Tests

Concentrate Constituent	H-1, Pct	H-2, Pct	M-1, Pct	M-2, Pct	M-3, Pct
Fe	61.90	66.45	60.35	67.20	67.70
SiO ₂	7.32	2.51	8.98	3.54	4.57
Al ₂ O ₃	1.93	0.82	4.42	1.13	0.23
CaO	0.27	0.29	0.30	0.45	0.32
MgO	0.51	0.27	0.29	0.50	0.41
Mn	0.18	0.10	0.25	0.28	0.27
P	0.074	0.005	0.033	0.016	0.010
S	0.055	0.209	0.357	0.113	0.009
Mesh Size	Screen Analysis				
65	0.7	0.5	4.1	0.2	1.1
100	3.4	0.7	6.7	0.3	0.3
150	8.9	1.3	12.0	0.6	1.3
200	14.3	5.2	11.5	1.8	3.4
325	15.2	12.8	10.2	10.1	8.8
400	21.0	18.6	12.8	16.8	23.5
800	21.1	37.8	18.5	38.2	26.9
1600	10.1	13.4	12.8	17.8	18.0
-1600	5.3	9.7	11.4	14.2	16.7
-325 mesh after regrind	87.4	92.0	78.0		

virtue of particle size and crystal chemistry, montmorillonite clays, or bentonites, aid the balling phenomenon, the green strength, the dry strength, and in some instances the fired strength. The coherency imparted to green pellets by bentonite materially reduces spalling and dusting of pellets during the



Figs. 9 to 12—Strength temperature relationships of iron ore pellets.

Table V. Strength-Temperature Relationships of Laboratory Fired Iron Ore Pellets

Additives, Lb Per Ton Concentrate Condition Firing Temperature	H-1				H-2		M-1		M-2		M-3
	10 b	10 b	10 b	10 b	10 b	10 b	10 b	10 b	10 b	10 b	10 b
	50 ls	50 ls							10 bx		
		MR	RG			RG		RG			
Crushing Strength, Lb											
200°C	20	20	20	20	10	10	20	20	10	10	20
500	30	30	40	60	30	30	130	180	60	20	130
900	120	90	100	110	50	100	180	370	160	50	280
1100	130	140	110	140	240	230	460	640	200	70	470
1200	580	470	190	270	650	780	670	700	330	270	680
1300	1800	1620	770	570	1220	1000	1450	760	930	650	880

high temperatures with Fe_2O_3 and SiO_2 to form readily fusible calcium ferrite and solid solutions of CaO , SiO_2 , and Fe_2O_3 . Borax functioned as a low temperature fluxing agent and probably aided the grain growth mechanism by a solution-precipitation phenomenon.¹¹ The pronounced beneficial effect of these additives is shown in Table III. H-1 pellets containing 2½ pct limestone and M-2 pellets containing ½ pct borax had remarkable strength properties in contrast to properties of the respective pellets without these additives.

The effects of firing temperatures on the crushing strengths of various kinds of pellets are given in Table V and are graphically portrayed in Figs. 5 to 12. It was observed that pellets of natural magnetite ore, such as M-1 and M-3 of Figs. 7 and 8, increased in strength with increasing temperatures, and at 500°C the pellets had crushing strengths of about 150 lb. These phenomena were not observed with pellets of artificial magnetite, such as M-2 and converted H-1 of Figs. 6 and 12, and were not observed with hematite pellets of H-1 and H-2 of Figs. 5 and 9. The low temperature cementation of natural magnetites has been discussed by Cooke and Ban.⁸ It was not known why this same phenomenon did not occur with pellets of artificial magnetite. Zetterstrom¹² has shown that artificial magnetite oxidizes at much

lower temperatures than natural magnetite and perhaps this rapid oxidizing nature of artificial magnetite inhibits cementation at these temperatures. In that low temperature cementation was coincidental with the exothermic phase change of natural magnetite to hematite, it was understandable that hematite pellets did not cement at low temperatures.

The strength-temperature curves of Figs. 5-12 give an indication of how the different kinds of iron ore pellets can withstand temperature gradients within a shaft furnace. If an arbitrary crushing strength of 400 lb is established for the minimum strength of acceptable pellets, it can be seen that the respective ores will withstand the following temperature differentials from 1300°C: M-3, 250°C; M-1, 225°C; H-2, 150°C; H-1 and M-2, 50°C. Pellet additives change these tolerable temperature differentials. For instance, 2½ pct limestone in the H-1 pellets increases the temperature differential from 50° to 125°C, and ½ pct borax in the M-2 pellets increases the temperature differential from 50° to 100°C. Three primary characteristics of the strength-temperature curves denote the relative response of the concentrates toward pelletization. These are as follows: the maximum strength that can be acquired, the extent of the low temperature strength increase, and the tolerable temperature gradient. These three factors are roughly composited in a diagrammatic sense by the area directly under the strength-temperature curves bounded by the 0-lb strength ordinate and the 1300°C temperature ordinate, see Figs. 13 and 14. With this notation it appeared that a classification of ores could be made in order of response toward pelletization. These would be as follows for the five ores studied: M-1, M-3, H-1 plus 2½ pct limestone, H-2, M-2 plus ½ pct borax, H-1, and M-2. This classification does not include the factors of abrasion and impact resistance or the important technical differences of shaft furnace pelletization of hematite and magnetite mentioned previously.

Table III summarizes the physical properties of pellets fired in the laboratory apparatus to an ultimate temperature of 1300°C. This table shows the effect of additives such as limestone and borax and the effect of the condition of the concentrates, such as that of reground or magnetically converted concentrates. The physical properties included are as follows: 1—pellet sizes, 2—pellet weights, 3—screen analyses of pellets after the drop test, 4—values of the crush, drop, and abrasion tests, and 5—the apparent density of the pellets. The determination of the apparent densities was made by use of an improvised widemouthed pynonometer.

The effect of regrinding concentrates prior to pelletizing appeared to be of small consequence. Reground ores of H-1, H-2, and M-1 produced pellets with greater abrasion resistance, and in the case of H-1 only slightly greater crushing and impact resistance. Reground ores of M-1 and H-2 yielded pel-

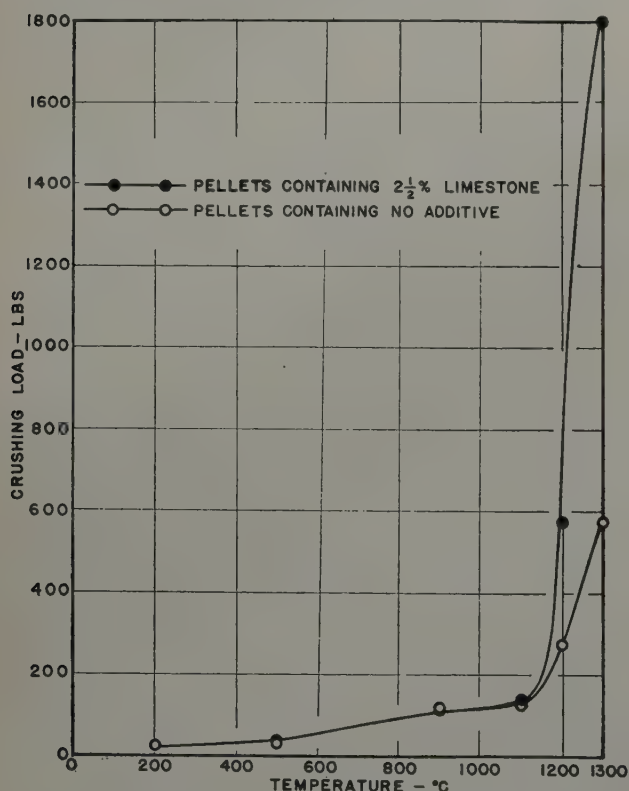


Fig. 13—Strength-temperature relationships of H-1 iron ore pellets showing the effect of pulverized limestone as an additive.

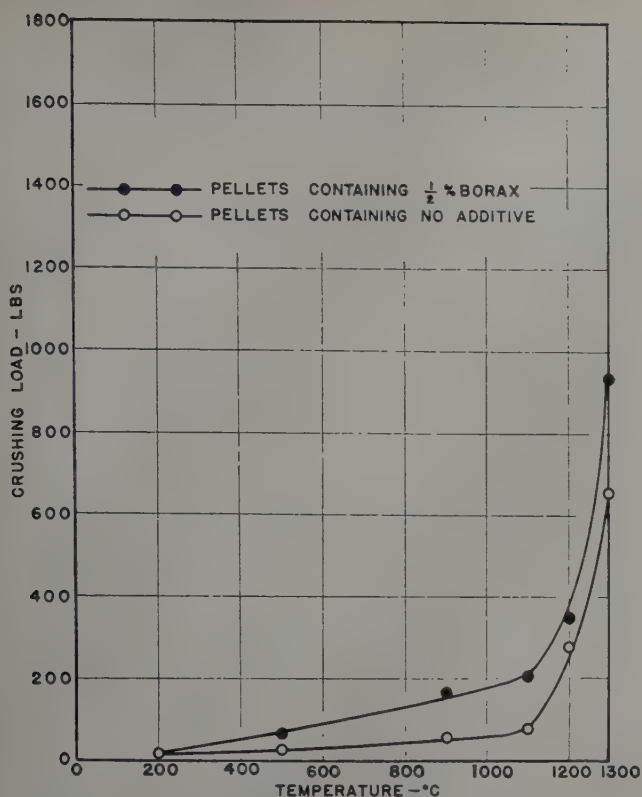


Fig. 14—Strength-temperature relationships of M-2 iron ore pellets showing the effect of pulverized borax as an additive.

lets with lower impact and crushing resistance. These two reground ores, M-1 and H-2, showed great tendency to spall while being fired, that is, the ore compacted too tightly in the wet stage, and the expulsion of balling water was inhibited during the firing process. Approximately 21 pct of the pellets composed of reground M-1, 9 pct composed of reground H-2, and 5.2 pct composed of 50 pct reground minus 50 pct original M-1 spalled during the drying stage of the firing process. No spalling occurred during the firing tests of all other pellets.

The beneficial effect of 2½ pct limestone in H-1 pellets and the less remarkable beneficial effect of borax within M-2 pellets are shown by the strength properties given in Table III. Visual examination of the cracked fragments of these pellets denoted conchoidal glassy fractures which exemplified the bonding action from the additives, especially so when contrasted to the rough crystalline fractures of the same pellets containing no additives.

The apparent detrimental effect of internal carbonaceous matter in iron ore pellets was evident from the decreasing strength properties of H-2 pellets containing increasing quantities of internal coal. Pellets containing 2 pct coal on the exterior, however, appeared to have greater strength properties than pellets containing lesser amounts of coal internally. This demonstration indicated that within pellets containing internal coal, gaseous diffusion lagged behind heat transfer, and carbon abstracted oxygen from the ore rather than from the hot gases. This reaction caused direct reduction of the iron ore and consequent defects in the pellet structure, such as hollow cores or contracted, sintered volumes from the phase changes at the very high pelletizing temperature. The reduced products reoxidized in some instances to the higher oxide states of iron, but left evidence of past history as a poor pellet structure with a low materials strength.

Provisions were made for better gas-solid reactions during the pelletizing tests by the application of coal to the pellet exterior. This naturally prevented disruption of the pellet interior, and served as a practical means of conveying fuel into the pelletizing zone of the laboratory apparatus.

Pellets comprised of H-1 converted to artificial magnetite and mixed with 2½ pct limestone were comparable in all strength properties to pellets of unconverted H-1 ore containing the same quantity of limestone. From this conversion some of the chemical advantages of magnetite were obtained without detriment to the pelletizing process. The commercial production costs of pelletizing hematite will unquestionably be greater than that of pelletizing magnetite because of the intrinsic source of thermal energy within magnetite. This cost difference, if appreciable, could dictate the necessity of beneficiating some fine-grained hematite ores by the magnetic roasting process.

Summary

This paper describes the design and operation of a laboratory apparatus for pelletizing iron ore by simulation of shaft furnace firing conditions. Strength-temperature relationships were determined for five different kinds of iron ore concentrates representing products from various regions planning to beneficiate iron ore by agglomeration. In addition to these relationships, the effects of various chemical additives and various methods of concentrate preparation were investigated. All the investigations were discussed with reference to the crushing, abrasion, and impact resistance of the pelletized products.

Acknowledgement

The authors wish to express their appreciation to the Cleveland-Cliffs Iron Co. for the support of this investigation and also for permission to publish the results.

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Mine Drainage at Eureka Corp. Ltd., Eureka, Nev.

by George W. Mitchell

THE property of Eureka Corp. Ltd. is located in the approximate geographic center of Nevada, 2 miles from Eureka, the county seat. The great sources of power, the Colorado, Snake, and Salmon Rivers and the rivers of northern California, are 300 to 500 miles distant, and no lines serve areas closer than 150 miles. Fuel for diesel and steam generation is available in Utah, 300 to 400 miles to the east. Eureka's railhead is 80 miles north where two trunk lines cross the county. A spur line serves Ely, 77 miles east. Good highways connect Eureka to the railheads.

Activity in the Eureka mining district began in the early 1870's. The oxidized high grade lead-silver-gold ore terminated against the footwall of the Ruby Hill fault, and in 1890 the main operations ceased. In 1938 Eureka Corp. Ltd. discovered ore in the hanging wall of the fault by diamond drilling.

The history of Eureka in the late 1800's indicates that there was some water at 600 to 800 ft in the old workings, probably accumulations above the water table which did not seriously interfere with mining operations. Both the Locan and Richmond shafts were sunk to a level below the table, but apparently the only serious difficulty with water occurred in the Locan. The steam pump used when the last work was done on the 1200 level in 1923, many years after exhaustion of the main orebodies, is still installed on the Locan 900 level. The capacity was about 500 gpm, lifting 750 ft to the 100 level, which connected with the surface. In addition to this, bailers were used to keep the 1200 level free of water. It is said that pumping in 1923 lowered the water in the Holly shaft, about a mile and a half away, but this seems doubtful. The pumping was of short duration because no ore was found.

When work at the new Fad shaft was started in 1941 Eureka Corp. Ltd. engineers were fully aware of the probability of encountering water in large volume. Their primary exploration and development had to be carried on at the 2250 level.

The first water was encountered at 300 ft. This was undoubtedly surface drainage in the bedding of the Pogonip limestone and was less than 100 gpm. The fractured, loose Hamburg dolomite at the water table was not well cemented, and relatively little water, 300 gpm, percolated through it with difficulty. At 1350 ft well-cemented dolomite containing some open fractures was encountered. These fractures

produced the first water of consequence, 750 gpm. At 1700 ft the volume was 1000 gpm increasing to the maximum during shaft sinking, 1600 gpm, at the 2000 level. Secret Canyon shale, a dry formation, was entered at 2100 ft, where a concrete water ring was placed to catch all of the water. The volume decreased rapidly to a constant flow of 1200 gpm. Below 2100 ft the shaft and stations remained in the shale and water was not a problem.

Several faults of moderate displacement, including the reverse Martin fault, had been intersected during the traversing of 1000 ft of wet Hamburg, but no undue quantities of water were encountered. Observations in the diamond drill holes in the ore zone area showed a rapid lowering of the water table.

The shaft was flooded when it left the dry shale and entered the water-bearing Eldorado dolomite on the 2250 level, crossing a fissure which paralleled the Martin fault. High pressure water doubled the volume then being pumped. Pipe failure through a water door bulkhead was a contributing factor.

Immediately following this flooding in March 1948 preparations were made to recover the shaft as rapidly as possible by increasing power and pump capacities as needed. Measurements before flooding indicated the water could be lowered at a fast rate. However, the water table did not recede as rapidly as expected and volumes required to lower the water in the shaft were higher. Obviously the size of the main water channel on the 2250 level was increasing because of erosion, allowing greater volumes to enter the workings and draining beyond the cone originally being drained during shaft sinking. Eroded material was being deposited in the shaft below the 2250 level in serious proportions. In December 1948 a second flooding of the Fad shaft was allowed for the purpose of reassessing existing conditions and studying alternate methods of attack.

The detailed geology of the Eureka mining district, see Fig. 1, has been described during the past 75 years by many geologists.¹ Only the general features and those which seem to affect the drainage problem will be discussed.

The old ore zone, mined between 1870 and 1890, is located in a wedge-shaped block of Eldorado dolomite between the footwall of the Ruby Hill fault and the underlying Prospect Mountain quartzite, see Fig. 2. Production of high grade oxidized lead ore containing high values in gold and silver has been variously estimated at \$50 to \$90 million. The tonnage mined was probably close to 1,500,000, nearly all of which was found above the water table.

The new ore, discovered by diamond drilling in the hanging wall of the Ruby Hill fault, is a flat-

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Discussion on this paper, TP 3578A, may be sent (2 copies) to AIME before Oct. 31, 1953. Manuscript, Sept. 22, 1952. Revised April 7, 1953. Los Angeles Meeting, February 1953.

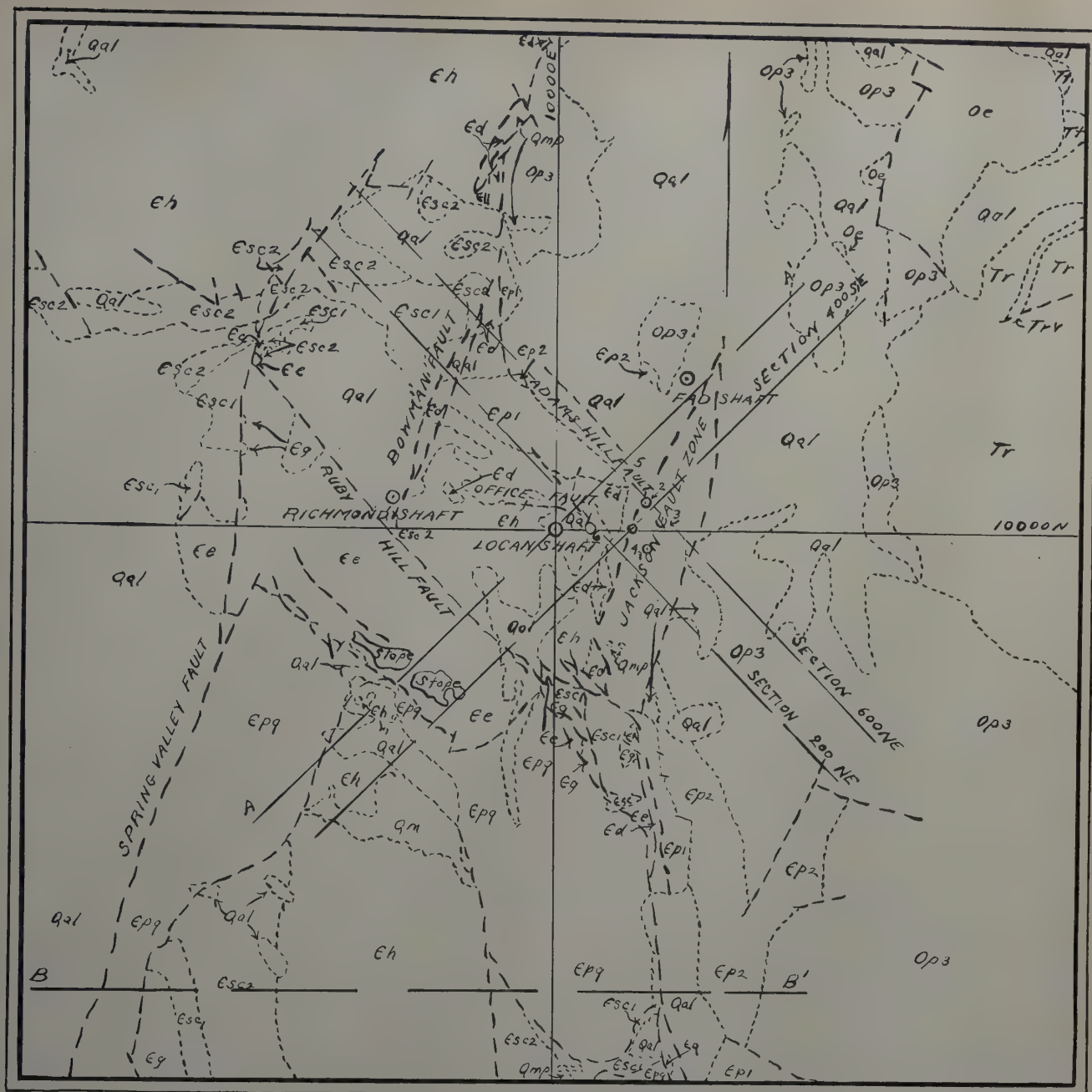


Fig. 1—Surface geology of Eureka mining district.

○ epg —	Prospect Mountain quartzite	○ Qm —	Quartz monzonite
○ Ee —	Eldorado dolomite	○ Qmp —	Quartz monzonite porphyry
○ Eg —	Geddes limestone	○ Tai —	Andesite (intrusive)
○ Esc1 —	Secret Canyon shale	○ Tae —	Andesite (extrusive)
○ Esc2 —	Secret Canyon shale	○ Tr —	Rhyolite
○ Eh —	Hamburg dolomite	○ Trv —	Rhyolite vitrophyre
○ Ed —	Dunderberg shale	○ Tt —	Tuff
○ Ep1 —	(Lower)	○ Tb —	Basalt
○ Ep2 —	(Middle) Pogonip limestone		
○ Op3 —	(Upper)		
○ Qe —	Eureka quartzite		
○ Qg1 —	Alluvium		

--- Fault
 --- Contact
 --- Section
 ○ Drill Hole

lying sulphide deposit of excellent grade lead and zinc with good gold and silver values.² Diamond drilling has indicated a limited tonnage of ore but geological conditions are such that tonnages could be large.

All of the sediments are folded and have been disturbed by thrust and normal faulting, in some cases of great magnitude. Thrust faulting from the west is a principal structural feature. It has elevated

and erosion has exposed the Cambrian beds which form the Prospect range, highest point 9600 ft above sea level, overturning the beds for the 8-mile length of the core of the range. Normal faults are numerous in the main part of the range. The Jackson fault and its branches on the east flank have been traced for several miles from the south end of the Prospect range to the north slope adjacent to the Fad shaft. On the west flank of the range the Spring Valley

fault, roughly paralleling the Jackson and less than a mile away, shows much displacement and, in places, the fault zone is wide and badly brecciated. Both the Jackson and the Spring Valley with their connecting cross faults could be conduits bringing water from the south toward the areas to be drained. The Jackson is known to be filled with intrusive rhyolite in some locations, and this condition, if sufficiently continuous, could form a partial barrier.

The geological column as exposed in the Fad shaft and drill holes is shown in Table I.

Table I. Geological Column Exposed in the Fad Shaft

Strata	Era	Level
Alluvium and surface rhyolites	Recent	100
Pogonip limestone	Ordovician and Upper Cambrian	690
Dunderberg shale	Upper Cambrian	250
Hamburg dolomite	Upper Cambrian	1060
Secret Canyon shale	Middle Cambrian	315
Geddes limestone	Middle Cambrian	0 to 200
Eldorado dolomite (formation in which the ore occurs)	Middle and Lower Cambrian	800 to 1000
Prospect Mountain quartzite	Lower Cambrian	300 to 1600

The three competent formations, the Prospect Mountain quartzite, Eldorado dolomite, and Hamburg dolomite, were badly crushed in the thrusting, the dolomites often recemented loosely. The less competent formations were drag-folded and distorted but retained their bedding.

Work done in the shales shows them to be all but impervious to water. Their dip places them between the mine workings and Diamond Valley, into which pumped water flows, where the water table is less than 100 ft below the surface. The shales should form a protective barrier, which may be imperfect, however, because they are cut by a number of faults.

No work has been done below the water table in the Geddes limestone, but upper exposures indicate that water could move easily along bedding planes.

The Hamburg and Eldorado dolomites are of identical chemical and physical character. Both contain a stockwork of fissures and faults, some of which are practically open channels. Both have been observed as capable of releasing quantities of water to mine openings. Experience indicates that there are large solution voids above, at, and just below the ground water level, but it is believed that only a few occur of sufficient magnitude to cause difficulty at the ore horizon 1200 ft below. The ore is within the Eldorado, and the Hamburg lies only a short distance above. There seems no choice but to drain these formations if the orebodies are to be successfully exploited.

The Fad shaft is 8x23 ft in rock section, containing four compartments in line. No. 3 compartment contains ventilation lines, one 12-in. and two 10-in. pump columns, and other pipe lines. No. 4, the manway, contains armored electric cable. The upper section, below the 30-ft concrete collar, is timbered for 300 ft and the next 250 ft solid-lined with concrete. The remainder consists of 6x6-in. H-section steel sets. Guides are surfaced fir timber, fastened to the sets with steel brackets. All shaft and pump stations are timbered with 8x8-in. H-section steel.

Standardization of equipment and extreme flexibility govern the general arrangement for handling water in the Fad shaft. The capacity of the equipment is relatively small, a number of units being used, so that shutdowns for repairs on any single

unit affect the general operation very little. Standard, well-tested equipment has been easier to obtain and cheaper in initial cost. Since replacement parts are readily available, inventory can be kept at a minimum.

To fit into these requirements, permanent pump stations, see Fig. 3, have been established at the 800 level, above the water table, see Figs. 4 and 5, at the 1700 level, and at the 2250. A temporary intermediate station was established at the 1200 because the sinker pumps could not lift the full distance from the 1700 to the 800 level.

The capacity of each shaft pump column is about 3000 gpm and friction losses are low. Standard weight pipe is used in the lifts. In the event of trouble in any one column the water may be diverted to the other columns by means of a valved manifold at the lower end. Valves are of heavier than standard design, but no extremely high head fittings are necessary.

Heavy lead-covered armored power lines direct from the power house circuit breakers to the pump stations are of sufficient capacity to operate two pumps. Meters on the power house distribution panel for each line record the demand, kilowatt-hour consumption, and operating time of two station pumps. Sinker and the submersible-type pumps also have their own panels. Underground transformers are necessary only to transform the power for the sinker pumps and auxiliary mine equipment, all other large motors being of the same voltage as the power generated.

The first three generating units were of 300-kw capacity. Three heavy-duty units of 1000-kw (1250-kva) capacity, generating 3-phase 60-cycle 2400-v power have been added, see Fig. 7. During the period in 1948 when recovery operations were in progress, after the first flooding of the shaft, four readily available lighter duty units of similar capacity were obtained. At the present time, August 1952, the 300-kw machines, because of size, have been taken out of service. Two more heavy duty units are ready for delivery. Any one of the large units may be shut down for extended periods without affecting progress. Two may be taken from the line for shorter periods without serious consequences.

A 600-hp double-drum hoist has been selected for the two hoisting compartments, while a small 50-hp single-drum hoist operates in No. 3 compartment. Three small compressors furnish adequate compressed air and a well equipped shop supplies construction and repair facilities.³

The submersible deep well pumps used are very well adapted for use in the shaft because of the small horizontal size of both the motor and pump, designed for 20-in. well casing. Three of the submersibles are driven by 450-hp, 3-phase, 60-cycle, 2400-v motors while the fourth has a 350-hp motor. The pumps are of 13 stages. The rated capacity of each of the three is 1600 gpm against 1000 ft, while the fourth is 1250 gpm, although all will pump greater volumes under lower heads. Two more 350-hp, 5-stage submersibles rated 2000 gpm against a 600-ft head are available. A higher head pump may be connected to these motors when needed. Each of the first four is hung on a 10-in. standard pump column from a location just above the water table, 1025 ft below the shaft collar on special steel supports. The approximate weight of each pump, column, and water within is 25 tons. Special long thread screwed joints are used on the upper end of

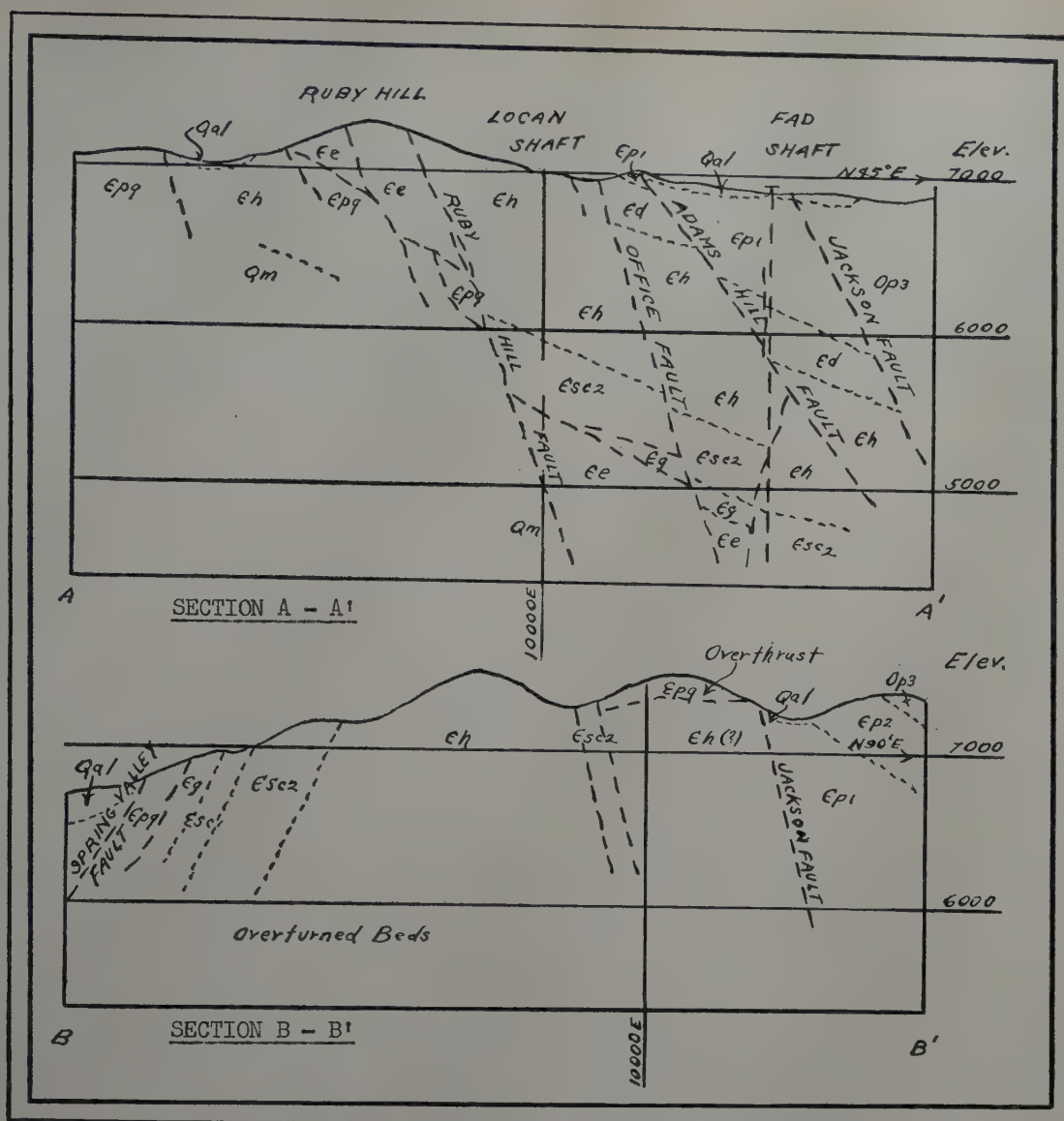


Fig. 2—Old and new ore zones of Eureka mining district.

the column. The pump and column is assembled and placed by use of a special sheave block arrangement using the shaft hoist as motive power. Two of the pumps are in No. 2 hoisting compartment and two in No. 3. No. 1 compartment is kept clear for the movement of men and supplies and No. 2 may be used for these purposes above the pump support.

Station pumps are all of the 3550-rpm, two-stage, horizontal type. On the 800-level pump station they are equipped with 350-hp, 2400-v motors, each delivering 1000 gpm of water to the surface. From the 1700 level to the 800 they are equipped with an impeller of slightly larger diameter and a 450-hp motor. Each pump has a capacity of 1200 gpm. From the 2250 level to the 1700 the impeller size is the smallest and the driver is a 250-hp motor. The horizontal-type station pump on the 2250 level will be replaced with submersible pumps and motors when the level is recovered, so that possible flash floods in the future may not cause loss of the station. Ten station pumps, installed on the two upper stations, were sufficient in late 1948 to pump 9000 gpm leaving two spares on the 1700 and one on the 800. Replacement of rotating elements, when necessary, was easily and rapidly accomplished.

All station pumps are automatically controlled by an electronic device developed in 1947.⁴ The probes are placed in the sump in such a manner that the water level in the sump makes and breaks contact with them. Each pump in succession comes into operation, if the water in the sump continues to rise, until a sufficient number are operating to cause it to recede. The last pump to come into operation cuts on and off automatically, keeping the sump at a predetermined level. Thus, only the number of pumps necessary to maintain a constant sump level are operated and, if the supply of water to the sump is cut off, all pumps will successively and automatically be shut down. The rare failure of the electronic device on any one pump, usually because of a burned-out tube, does not affect the other pumps. It merely keeps the pump out of operation until repairs are made. An alarm installed in the power house panel reveals such failures immediately and no full time attendant is necessary at any of the stations. The panel records are checked daily, particularly for a dropping off in the demand of each pump motor. This is a good indicator of pump impeller ring and interstage bushing wear. The pump inspector is equipped with a clampon ammeter so

he may test each motor for demand as station inspections are made.

All sumps are at a higher elevation than the pump stations so that all station pumps may have a positive suction head. No difficulty has been experienced in keeping the pumps primed, but a precaution against dry pumps has been provided. If the discharge of any pump fails to come up to a predetermined pressure in a few seconds after starting, it is automatically shut down by a pressure governor installed in the starter and an alarm is sounded in the power house panel. Additionally if, during operation, the pressure is reduced because of a broken pipe line or other similar cause, the pump is shut down by the pressure governor.

Station pump wear has been low for two reasons. First, the sediment in the water, when there is any, is non-abrasive. All of it has been limestone, dolomite, and shale particles containing little or no quartz or other hard material. The fine material is quite difficult to settle. Second, the coarser gritty particles are settled out in settling ponds at the sump discharge end or in the sumps themselves. The settling ponds designed into each sump have proved too small to remove effectively a large amount of coarse solids and keep the main sump free of them. Erosion in the 2250 level water channel in 1948 produced an excessive amount of solids which built up in both the settling ponds and the sumps. However, in normal operation it is believed that the greater part of the coarse solids will be settled out in the ponds and easily removed.

The water is slightly alkaline, and though it is hard, no lime has been deposited in the pipe lines or pumps. Corrosion is not likely to be a serious problem, even after the sulphide orebodies are being mined, because of the neutralizing effect of the dolomite rocks.

The heat generated by the large motors in the pump stations has been a problem and will remain so until a second opening providing adequate ventilation has been made. It is not economically feasible to blow sufficient air to the stations, although two 18-in. steel ventilation lines are in the shaft. However, the 60° water provides a cool medium for heat exchange. Two large radiators which circulate mine water and are equipped with large propeller-type fans have been placed on the 1700 level pump station. They absorb sufficient heat to maintain proper station temperature.



Fig. 3—No. 2 section, 800 level pump station, pump side. At right is the 1700 level pump discharge line and the 800 level suction lines. At center is the 800 level discharge line. Motors and starters are at left of center posts.

Many engineers whose advice has been sought suspect that the water course, intersected on the 2250 level, may be one of a few carrying large quantities which might be grouted, frozen, or otherwise blocked. Although this water course can be grouted, however, it seems certain that many similar channels, duplicating the condition now existing, will be cut in developing the ore zone. The quick response of the water level, as measured in the diamond drill holes, to any change of level in the Fad shaft indicates a close hydraulic connection between the two areas, although they are 1200 ft apart. Blocking off one channel will merely cause the water to find a new way into the workings, but it seems likely that grouting in localized areas will be used. Most engineers, including those of the U. S. Geological Survey Ground Water Division, believe that a typical ground water problem is presented. Below the ground water level porous rock with interconnecting movement, bedding, and jointing planes are saturated with water. The same rocks are the host rocks of the ore. The expense of lowering the water table in the vicinity of the workings to a safe horizon will be great, but this appears to be the only solution to the problem.

The general topography immediately suggests the possibility of a drainage tunnel. Although the shaft collar is approximately 7000 ft above sea level, all the valleys within a radius of 50 miles are about 6000 ft. No tunnel of reasonable length could tap the water table in the vicinity of the ore zone, but some 700 to 800 ft of head could be removed from the total lift by a tunnel about 2 miles in length. Ultimately the economics of such a tunnel will be worked out, but at the present time such a project does not seem feasible.

The reduction in the horse power required to lift water and friction losses in pump columns and electrical and pump efficiencies has received much thought. Air and gas lifts have been considered, by themselves and in combination with conventional pumps. While there are possibilities of large savings in power through increased efficiency, the engineer is always faced with the fact that the energy necessary to lift a given weight of water a given distance is constant.

The plan to unwater consists of five phases: 1—draw down and recovery tests, 2—pumping from the Fad shaft with recovery of the 1700 level as the first objective, 3—development of additional water to bring the total volume to 12,000 gpm, 4—continued pumping to lower the water table to a safe horizon, and 5—recovery of the 2000 and 2250 levels, to be advanced to the ore zone and development of the ore.

The draw down test will require about a month. The volume pumped during the draw down will be 3600 gpm, the capacity under an 800-ft head of two of the three 450-hp submersible pumps. One of the 450-hp submersibles will be a standby so a constant volume can be maintained, the allowable maximum variation being 3 pct. The volume will be controlled by an orifice volume measuring device and throttling valves in the submersible pump discharge line, all of which will be located on the 800-level station. The volume pumped from the shaft will be delivered to the 800-level sump and four station pumps will lift to the surface. Three of the station pumps will operate continuously and the fourth intermittently to handle the 3600 gpm. Stand-by station pumps are provided.

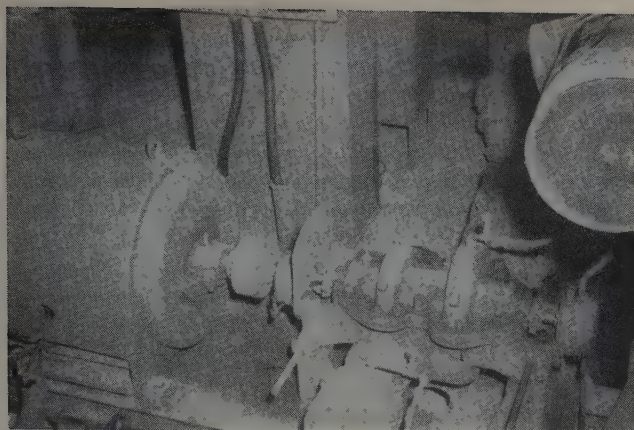


Fig. 4—No. 1 pump unit, 350-hp, in No. 1 section, 800 level pump station.

Installed in the main discharge line on the surface is a Parshall flume, of 14,000-gpm capacity, equipped with a recording volume meter. After being measured the water will be discharged through 5700 ft of 24-in. steel pipe to an open ditch. Another Parshall flume of the same dimensions as the first is to be installed in the stream $2\frac{1}{2}$ miles from the shaft to check the volume for return circulation to the mine. Plans are to check the volume again at a point 4 miles from the shaft if no recirculation is observed closer.

Recording devices at the diamond drill holes and the Holly shaft will observe the ground water level as it recedes.

Approximately 175,000,000 gal will be pumped during the test. It is expected that this volume will lower the water in the shaft about 450 ft, in the diamond drill holes 250 ft and in the Holly shaft, $1\frac{1}{4}$ miles distant, 1 or 2 ft. The shape of the curve of draw down, irregularities in flattening or steepening, will indicate any additional water from lines of recharge or the existence of good barriers. It is expected that the most economical time-volume figures may be calculated from these observations.

If it appears necessary, these figures will be checked by allowing the water to recharge for a period equal to, or possibly shorter than, the draw down. A good estimate can then be made of the recharge to the basin formed after the water table has been lowered to the mining horizon.

The initial pumping rate following completion of the draw down test is expected to be approximately 6000 gpm. This or a slightly lesser or greater volume will be maintained, depending upon the volume which may be taken without again causing erosion in the 2250 level channel. At this volume it is expected that the 1600 horizon will be reached rapidly and that a month or two of steady pumping will lower the water to the 1700 level. The head capacity of the four submersible pumps is sufficient to lift 6000 gpm from a point below 1700 to 800 level.

The contact between the overlying Hamburg dolomite and the Secret Canyon shale cuts through the 1700 shaft station, see vertical section. The dry shale extends approximately 600 ft to the south. A crosscut will be extended in this direction to a point near the bottom of the shale close to the underlying Geddes contact. With the protection of a water door in the crosscut and valved collar pipe, drill holes will be extended ahead of and below the face into the water-bearing Geddes and/or Eldorado. In this manner it is expected that additional water can be

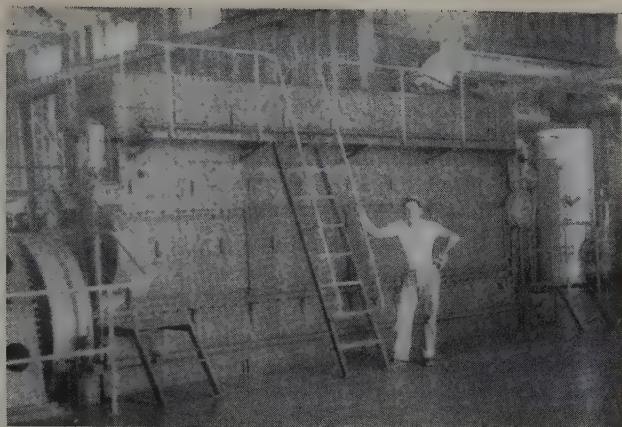


Fig. 5—Power unit No. 1, 1000 kw.

made, bringing the total volume pumped to approximately 12,000 gpm.

The objective is to increase volume to shorten time. Pumping from the Fad shaft and this area will draw down the water table above the ore zone, forming a trough-like basin in the location of the projected 2000 and 2250 levels crosscuts.

As the water table recedes and the drill holes begin to produce less water, the 1700 crosscut will be extended into the Geddes and Eldorado formations to maintain the volume. When practicable a winze above the ore zone will be sunk and possibly water wells drilled from the 1700 level for the dual purpose of making water and reaching the mineralized zone. Ultimately the 2000 and 2250 crosscuts will connect to the bottom of the winze while a connection from the top of the winze to the surface will provide ventilation and a secondary exit from the stoping area.

Addenda

The draw down and recovery test previously described was completed on January 31, 1953.

The predicted volume and time required to unwater to the 2250 level, as well as the volume to hold at that level, are considerably in excess of the figures estimated from the experiences of 1948. Decision to postpone the major unwatering program has been made, pending further study.

The cost of power and the practicality of developing sufficient water to bring the time factor within feasible limits are the main problems. Larger expenditures than originally anticipated will demand additional information as to the tonnage and grade of ore to be mined.

Acknowledgments

The writer wishes to thank Thayer Lindsley, President of Eureka Corp. Ltd., for permission to write this paper and John K. Brozo, Chief Engineer, for making maps and giving helpful criticism.

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An Investigation of the Collecting Effects Of Fatty Acids in Tall Oil on Oxide Minerals, Particularly on Ilmenite

A preliminary investigation of the collecting effects of various fatty acids in tall oil has led to a conclusion that the collecting power of fatty acids used in flotation increases with increasing unsaturation of the hydrocarbon chain. Linoleic acid was found to be an excellent collector for oxide minerals.

by R. T. Hukki and O. Vartiainen

TALL oil is a mixture of fatty and resin acids extracted as byproducts in the sulphate pulp industry. In the field of flotation a well-known example of this type of product is Reagent No. 708 of the American Cyanamid Co. Its successful use as a flotation collector with many nonsulphide minerals has been described by Falconer and Crawford.¹

The composition of crude tall oil varies within wide limits.^{2, 3} By a number of methods the crude oil may be refined to produce a variety of refined products.

This investigation presents a preliminary comparative study of collecting properties of the fatty acids in tall oil. In Finnish tall oils these constituents are primarily palmitic, oleic, linoleic, and linolenic acids.³ The first is a saturated fatty acid having 15 carbon atoms in its hydrocarbon chain, plus one in the carboxyl group. The others have 17-carbon-atom chains and are unsaturated. It is of

particular interest to note that palmitic, oleic, linoleic, and linolenic acids represent increasing degrees of unsaturation with 0, 1, 2, and 3 double bonds, respectively.

To test the collecting properties of the four fatty acids separately, the vacuum flotation technique described by Schuhmann and Prakash⁴ was used. By this method the boundary between flotation and non-flotation conditions was approximately delineated, with the use of ilmenite and several other minerals. Collector concentration and pH were the principal variables. On the basis of information obtained, actual flotation tests were run on an ilmenite-bearing ore in a laboratory flotation cell.

Through the courtesy of P. Kajanne from the Department of Chemistry, Finland Institute of Technology, the following reagents were obtained for this investigation: 1—Palmitic acid, of Judex, had a purity of 95 pct as determined by parallel iodometric and spectrophotometric measurements. 2—Oleic acid, product of May and Baker Ltd., indicated purity of 98 pct. By iodometric and spectrophotometric determinations the oleic acid content was found to be 83.8 pct. In addition, it contained 6 pct linoleic, 1 pct linolenic, and 9.2 pct saturated fatty acids.⁵ 3—Linoleic acid was prepared from soybean oil by low-temperature crystallization. Its linoleic acid content was 95 pct by the methods indicated above. 4—Linolenic acid was prepared from linseed oil by low temperature crystallization. Its linolenic

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This paper is based on a thesis by O. Vartiainen submitted in partial fulfillment of the requirements for the degree of B.Sc. in the Department of Mining Engineering, Finland Institute of Technology.

Discussion on this paper, TP 3542B, may be sent (2 copies) to AIME before Oct. 31, 1953. Manuscript, Nov. 26, 1952. Revised Feb. 27, 1953.

acid content was estimated by the two methods at only 65 pct, the balance being chiefly oleic and linoleic acids.

Palmitic acid was used as an aqueous suspension of potassium palmitate containing 5 pct ethanol. The others were used as such or in the form of sodium salt solutions.

In vacuum flotation tests, 100 to 200 mesh fractions of ilmenite (TiFeO_3) concentrate assaying 48.2 pct TiO_2 were used. The concentrate was prepared from Otanmäki ore by tabling and multiple cleaning of the table concentrate by low and high intensity magnetic separation.

Minerals

Similar samples were also prepared of Otanmäki magnetite, Fe_3O_4 , and pyrite, FeS_2 . For a comparison, samples of Otanmäki hornblende with a certain amount of chlorite and biotite were tested as representatives of respective gangue silicate minerals. In addition, small samples of high grade hematite, Fe_2O_3 , and rutile, TiO_2 , of unknown origin were prepared as examples of oxides that are similar to ilmenite and magnetite.

One-gram samples of mineral, 100 ml of water of suitable pH, and the reagent under investigation were introduced into the thoroughly cleaned vacuum flotation cylinders, which were then closed with glass stoppers. For 25 min the cylinders were tumbled in the frame at a speed of 41 rpm. After this conditioning, glass stoppers were removed and rubber stoppers attached to an inserted vacuum line. Dissolved air was allowed to precipitate out under vacuum so that conditions of flotation and non-flotation might be observed. Full flotation was indicated by the accumulation of all mineral particles with ascending air bubbles at the air-water interface. Non-flotation was revealed by the total passivity of mineral particles toward air bubbles. In the latter case, all the particles were retained in the bottom of the cylinder.

Fig. 1 is typical of the results obtained by the Schuhmann-Prakash method. It shows the flotation conditions for ilmenite. The coordinates are respectively pH, or log hydrogen-ion concentration, and

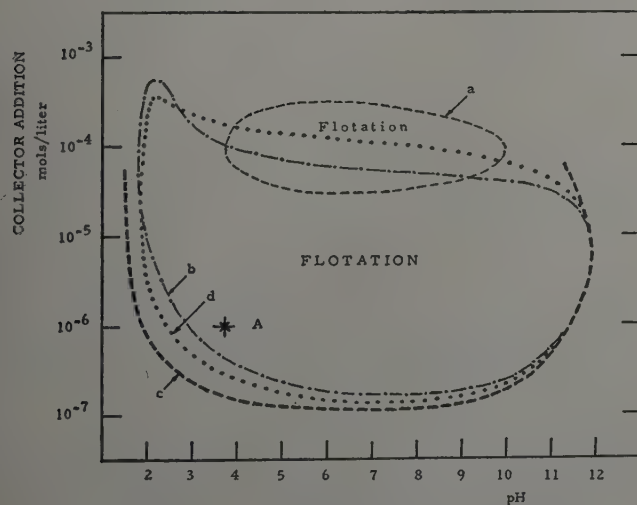


Fig. 1—Delineation of the experimental results of vacuum flotation tests on ilmenite with a—potassium palmitate, b—sodium oleate, c—sodium linoleate, and d—crude sodium linolenate. The graphs indicate the transfer boundaries between non-flotation and flotation, but do not show whether flotation is poor, good, or excellent. The quantitative behavior at point A of the linoleate and oleate soaps is shown by photographs in Figs. 2a and 2b.

log collector addition. This figure shows flotation to be possible inside the closed curves. The extreme boundaries for the linoleate reagent are not known. The position of the curves is approximate, since the Schuhmann-Prakash method gives semi-quantitative results only, but with great rapidity.

Results of Experimental Techniques

The observations of about 550 separate tests conducted by vacuum flotation method may be summarized as follows:

1—The four fatty acids act as collectors of ilmenite over a wide pH-range.

2—The collecting power of linoleic acid, or of the respective sodium soap, is superior to that of oleic acid, or of sodium oleate. This is clearly shown in Figs. 2a and 2b.

3—The collecting power of oleic acid is much stronger than that of palmitic acid.

4—The collecting power of the crude linolenic acid available to us is intermediate between that of linoleic and oleic acids.

5—The superiority of linoleic acid over oleic acid is equally pronounced with magnetite, hematite, and rutile as with ilmenite.

6—When linoleic acid is used as collector, the air bubbles appear completely armored with mineral particles, while in the other cases under similar experimental conditions only a part of bubble surface is covered with the mineral as shown by comparison of Figs. 2a and 2b.

7—All four fatty acids act as collectors of pyrite over roughly the same pH-range as in case of ilmenite.

8—The collecting power of the fatty acids on silicates, e.g. hornblende, is definitely inferior to that on metallic oxides. With the fatty acids under investigation, it appears to be in the order palmitic < linoleic < linolenic < oleic acid.

Discussion

These observations have led the authors to the following theorem: The collecting power of fatty acids used in flotation increases with increasing unsaturation of the hydrocarbon chain.

This is supported by tests conducted with palmitic, oleic, and linoleic acids. It is not supported by tests with linolenic acid. The sample of linolenic acid used, however, contained only 65 pct linolenic acid. Consequently results cannot be regarded as proofs against the above theorem.

In this connection it is of interest to compare the investigations of Keck, Eggleston, and Lowry.⁶ They report that the collecting power on hematite of the fatty acids they tested was in the order of oleic > lauric > myristic > palmitic > caprylic acid. This order in no way contradicts the observations summarized in this paper since lauric, myristic, and caprylic acids represent saturated fatty acids. In the paper describing flotation of magnetite, Keck and Jasberg⁷ state that "unsaturated oleic acid with 18 carbon atoms was a better collector than saturated stearic acid, which also has 18 carbon atoms" and that with the saturated fatty acids flotation increased with the number of carbon atoms.

The observations made by the present writers indicate that selective separation of pyrite from heavy metal oxides with fatty acid collectors will be difficult without modifying reagents. On the other hand, separation of oxides from certain silicates appears possible without modifying reagents and over a considerable pH-range.

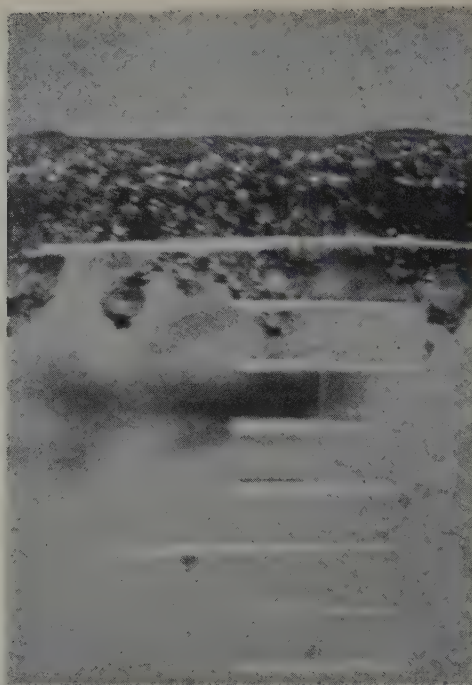
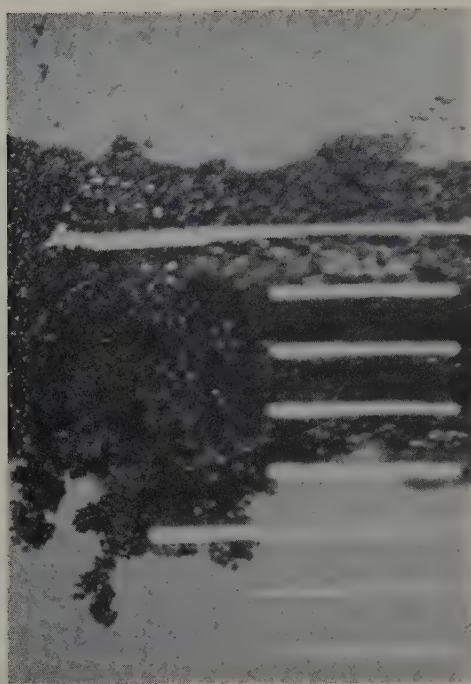


Fig. 2a—A photograph of vacuum flotation tests on ilmenite with linoleic acid, and Fig. 2b—with oleic acid. In these tests, 0.001 millimols of Na-linolate and an equal amount of Na-oleate were used at pH 3.7 for 1-g samples of 100 to 200 mesh ilmenite. Note how fully laden the tiny air bubbles are as shown in Fig. 2a, how completely covered the bubbles appear, and how thick the froth layer is when all particles are floated. In the test illustrated in Fig. 2b the coalescence of small bubbles has been less hindered and thus larger bubbles have formed. Each bubble is only partly covered with ilmenite, and the froth layer is relatively thin and carries only a fraction of the total sample.

In view of the fact that the sulphate cooking process effects well-known alkali isomerization phenomena in the double bond structure, it is important to note that further investigations are needed to reveal the collecting power of different linoleic acid isomers.

On the basis of the above work it was of interest to test the applicability of the described theorem in actual flotation tests. The general rule in flora is that the fatty acids contained by plants increase in degree of unsaturation with increasing latitude.³ Therefore samples of tall oil extracted from pines grown in northern Finland would have been highly desirable. At our request, several samples of such tall oils were submitted by Oulu Co. In the processing of crude tall oil, palmitic acid seems to concentrate to first fractions relatively low in resin acids, oleic acid in the middle fractions, and linoleic acid in the last fractions richer in resins. (Their resin content was 25 to 40 pct.)⁸ The amount of linolenic acid in the crude tall oil is low, of the order of 2 pct,⁸ and can be disregarded. The effect of resin acids in tall oils on flotation is somewhat obscure. Their collecting power in the pure state is unknown to the authors, but they seem to cause a certain amount of frothing.^{9, 10}

Although no exact analyses of the fatty acid distribution in the samples of tall oils submitted were available, those samples in which the linoleic acid content was highest gave the best results in the laboratory flotation experiments with Otanmäki ilmenite ore. As a matter of fact, one such tall oil reagent gave concentrates assaying 44 to 46.8 pct TiO_2 , the maximum being about 48.5 pct TiO_2 with this ore, representing recoveries of 75 to 79 pct out of a deslimed ore assaying 15 pct TiO_2 . About 10 pct increase in recovery would result in continuous practice from the retreatment of flotation middlings. These results are at least equal if not su-

perior to those obtained under the same experimental conditions with any other conventional reagent including many of foreign manufacture.

Conclusions

1—A number of oxide minerals of economic importance can be concentrated with tall oil collectors of domestic Finnish origin. 2—Tall oils rich in linoleic acid are superior to fatty acid collectors conventionally used. 3—New flotation reagents of superior collecting power can be developed for flotation of oxide minerals. 4—The development of a means of commercial production, at a reasonable cost of, for example, linoleic acid of sufficient purity and proper isomeric constitution, would be of worldwide interest in the flotation of oxide minerals.

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Sulphur Recovery from Low-Grade Surface Deposits

by Thomas P. Forbath

THE sudden realization that known sulphur reserves amenable to mining by the Frasch hot water process are nearing exhaustion focused attention on widely scattered surface deposits throughout the world. These deposits are not necessarily of lower sulphur content than ores located underneath Louisiana or Texas salt domes which usually average about 30 pct sulphur disseminated in limestone matrix. Their near surface occurrence, however, renders exploitation by the Frasch process impossible. As is well known, the Frasch process depends on the presence of 500 to 1000 ft of overburden and cap rock above the sulphur deposits to permit melting underground sulphur in place by diffusing hot water under pressures of 200 to 600 psig in the formation and raising the molten sulphur to surface by air lift. This process renders possible the production of pure sulphur which is 99.5 pct pure without any subsequent treatment.

Surface deposits contain sulphur in the same range of concentrations as the salt dome deposits, i.e., from 10 to 50 pct sulphur, associated with various gangue materials such as silica, limestone, and gypsum. The principal distinction, then, does not lie in the percentage of sulphur contained in the ore, but in the geological nature of the deposit.

A recent study¹ of the world sulphur supply situation estimated 1950 sulphur production in the free world countries at 5.6 million long tons, of which the United States produced 5.2 million tons, or 93 pct of the total. While America's domestic needs alone would have been covered by national production, about 1.4 million tons were exported during the same year. Despite all the steps which are being taken to restrict use of elemental sulphur and to force the fullest possible development of alternate sulphur sources here and abroad, the deficit in elemental sulphur production will probably increase with time. As a result of intensive prospecting for oil throughout the Gulf Coast area discovery of significant new salt domes is held unlikely. With the growing scarcity of sulphur and what appears to be an inevitable rise in price, recovery from deposits not amenable to Frasch-process mining assumes greater economic importance.

Untapped Reserves

The most important deposits in this category are located in Sicily, where elemental sulphur occurs in Miocene limestone and gypsum formation. Sulphur

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Discussion on this paper, TP 3628H, may be sent (2 copies) to AIME before Nov. 30, 1953.

content of these ores ranges from 12 to 50 pct with an estimated average of 26 pct. Although quantitative estimate of these reserves is not available it is held that they exceed 50 million tons of sulphur. Similar deposits occur also on the mainland which contribute about one-third of Italy's total current annual production of 230,000 tons, but these are known to be nearing exhaustion.

Significant surface deposits of volcanic origin are located in South America, Japan and western United States, silica being characteristic gangue constituent. The largest of these deposits are in South America. More than 100 extend over a zone 3000 miles long, paralleling the west coast of South America. Total sulphur content of these deposits has been estimated to be as high as 100 million tons. The main islands of Japan also possess at least 40 known volcanic sulphur deposits with probable reserves of 25 to 50 million tons.² Prospected reserves in western United States might amount to 2 million long tons, principal deposits being located in the northwestern part of Wyoming, southern Utah, and eastern California.

Volcanic deposits of lesser importance are found around the Mediterranean, in Turkey and Greece, and in Africa, Egypt, Abyssinia, and Somaliland.

Beneficiation Methods

Different methods of beneficiation have been used in these various locations. In Italy the Calcarone kiln and Gill regenerative furnaces are used exclusively. Both utilize heat liberated by burning part of the sulphur in the ore to liquify or vaporize the remaining sulphur, which is recovered by solidification or condensation. The Calcarone kiln is of conical shape, generally 35 ft in diam at base and 18 ft high. A kiln of 25,000 cu ft capacity burns for about two months and yields about 200 tons of sulphur. The Gill furnace consists of a series of chambers with domed roofs. Sulphur is burned and melted in one chamber at a time and the hot combustion gases are used to preheat the ore charge in the subsequent cell. These furnaces operate on a cycle of 4 to 8 days. The recovery yield of both systems is about 65 pct. Sulphur losses amount to 25 pct through the combustion to sulphur dioxide; about 10 pct is retained in discarded calcines. Ores containing less than 20 pct are not considered suitable as furnace feed.

These methods are not only wasteful because of the low recovery obtained, but represent a serious atmospheric pollution problem. Sulphur produced ranges from 96 to 99 pct purity and thus does not match Texas or Louisiana sulphur. Owing to the present shortage, sulphur in the Middle East sells

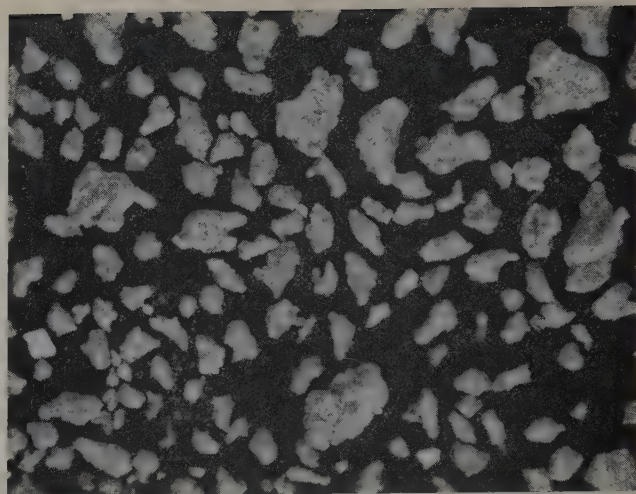
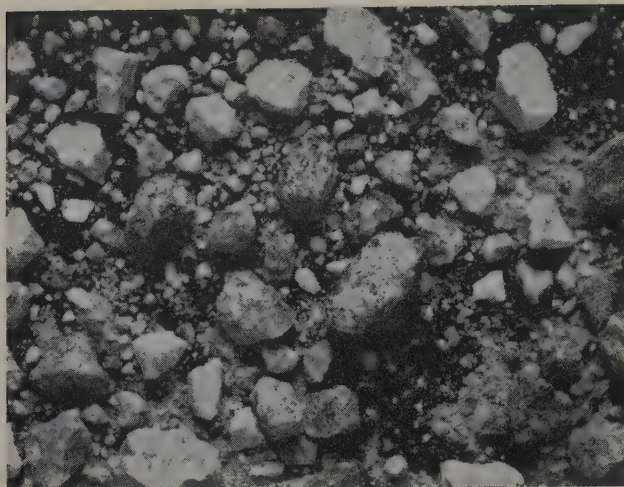


Fig. 1 (left)—Sulphur ore from Colombia, S. A. Even fine grinding does not liberate all the sulphur, which is thoroughly disseminated in the gangue. Fig. 2 (right)—Oversize particles of sulphur formed by coalescence of sulphur droplets after treatment in gangue separator. Both views X3.

for as high as \$200 per ton. In spite of this, the Italian sulphur industry is still beset with economic difficulties. These can be ascribed to high mining costs, averaging about \$8 per ton of ore, as well as to high recovery costs.

In South America the batch autoclaving method is widely used. In this process sulphur ore is crushed to about 2-in. size and a batch of approximately 1.5 tons is charged to cast iron autoclaves 4 ft in diam and 7 ft high containing a grate with $\frac{1}{4}$ -in. openings on which the charge rests. After charging, the autoclaves are bolted and live steam of 60-lb pressure is admitted for a heating period of 1 hr. At the end of this period, a valve at the bottom of the clave is opened and the molten sulphur which passed through the grate is drawn off into concrete pits where it solidifies. The autoclave is then opened and the leached ore removed and discarded. The complete cycle takes about 2 hr. Since the steamed ore still retains 20 to 30 pct sulphur, recovery yields are always very low. Although the purity of sulphur produced by this method is acceptable, production costs are excessive because of low recovery and high utility and labor costs caused by the batch-type operation. Even at the present time, when sulphur sells at \$100 and higher at South American ports, the autoclaving method offers a narrow margin of profit at best.

Several attempts have been reported on the application of straightforward froth flotation methods to recovery of sulphur from surface ores.^{3,4} While sulphur is considered to be a *natural floater*, it was usually found that production of a high purity concentrate with high recovery yields was not possible. This difficulty is largely due to the fact that in most ores sulphur is very finely disseminated within the gangue and even after fine grinding (100 pct through 200 mesh) microscopic examination shows a large proportion of sulphur and gangue particles cemented together. Using more extreme grinding renders the flotation step more involved. Finely ground sulphur flocculates very intensely and occludes particles of gangue. These occlusions are hard to break down.

Processes employing solvent extraction to recover elemental sulphur from ores have also been proposed and described in literature.⁵ The drawback of these processes usually lies in high vapor pressure, inflammability, and cost of solvents involved. While use of carbon bisulphide, for example, is technically feasible at sea level, CS_2 would behave as a gas at

the altitude of most of the South American deposits, and at the temperatures involved in a solvent extraction process.

These and other methods were used at one time or another at nearly all the known sulphur deposits throughout the world. None proved competitive with Gulf Coast sulphur.

The Chemico Process

For over 10 years, Chemical Construction Corp. has received inquiries from foreign countries regarding concentration of sulphur ores, and about 5 years ago decided to attempt development of a process which would yield high purity sulphur at an economical cost. The first ores tested under this program originated from Colombia, S. A., Egypt, and Milos Island, Greece. On each of these ores, recovery by flotation alone did not prove feasible, even with fine grinding to -200 mesh. The idea arose of separating sulphur from gangue by melting prior to flotation. Use of high boiling liquids at atmospheric pressure was first considered. Experiments in which ground sulphur ore was suspended in 98 pct sulphuric acid and heated above 240°F, i.e., the melting point of sulphur, under agitation, showed the way to further developments. It was observed that in relatively short times all the sulphur could be melted and that the molten sulphur droplets tended to coalesce and agglomerate. Also noted was the fact that following such thermal treatment, subsequent separation by flotation was relatively easy.

The idea of handling large amounts of hot sulphuric acid in commercial installations was not very attractive; consequently in the next step of the development program it was attempted to melt sulphur away from gangue by suspending the ground sulphur ore in water and heating the slurry under a pressure of 35 psig above the melting point of sulphur, while maintaining it in a state of agitation. It was again recorded that in relatively short periods, from 15 to 60 sec, all the sulphur could be physically separated from the gangue. At the end of the heating period cold water was injected into the agitated mixture and the temperature reduced to 185°F, freezing all liquid sulphur particles. Subsequent recovery of sulphur by froth flotation was again found to be relatively simple. These experiments provided the basis for the present Chemico sulphur recovery process, since thermal treatment in the gangue separator is considered the key to its suc-

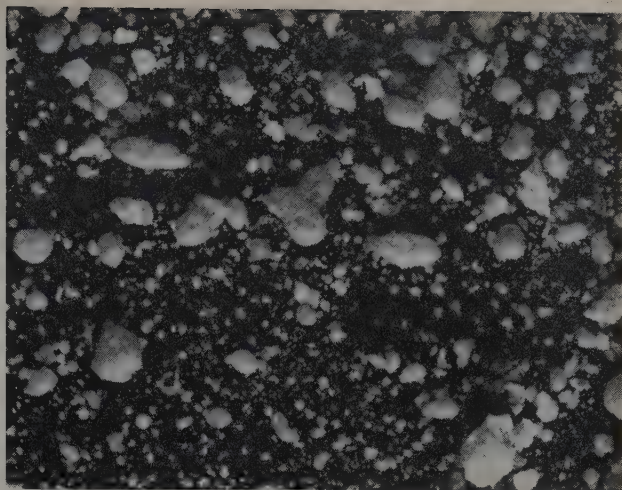
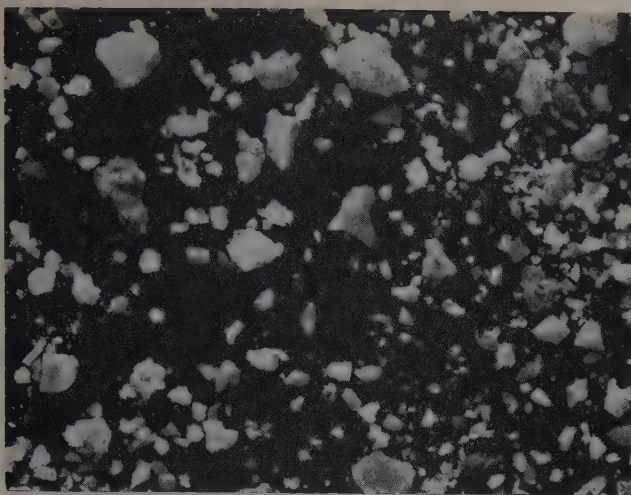


Fig. 3 (left)—Rougher flotation follows thermal treatment, yields this tailing containing 2 to 3 pct S. Fig. 4 (right)—Cleaner flotation cells produce a 90 to 95 pct S concentrate, and a middling which is recycled to rougher flotation. X15.

cessful application to a wide variety of highly refractory ores.⁶

To adapt the thermal treatment step to a continuous process a coil-type reactor was developed. Ground slurry was pumped through a pipe coil at velocity sufficient to provide turbulent flow conditions. Live steam of 35 psig pressure was injected into the coil, which was kept under a pressure of about 30 lb. The coil was made long enough to provide retention time of about 1½ min. At the tail end of the coil cold water was injected to quench molten sulphur particles and the cooled slurry was then discharged through a pressure relief valve.

It was found that the thermal treatment step obviated the necessity of fine grinding usually required for satisfactory separation by straight flotation alone. This means a considerable saving in power consumption, since power required to reduce a given amount of sulphur ore feed from 12-in. size to—200 mesh is roughly three times that required to go from 12 in. to—20 mesh.

Process Details

The Chemico process developed on the basis of the above considerations can be described as follows. Ore is crushed in primary and secondary crushers about ¾-in. The crushed ore, Fig. 1, is then fed to a pebble mill which grinds it —28 mesh. Since dry grinding of sulphur-bearing ores is inherently dangerous because of fire and explosion hazards, wet grinding is employed. Pulp concentration in the pebble mill is kept about 50 to 60 pct solids. The pebble mill is operated in a continuous closed cir-

cuit with a 28-mesh screen. The grinding step invariably causes disintegration of some 50 pct or more of the sulphur into particles of 1 to 5 micron size. The ground ore is next suspended in water to form a slurry of about 30 pct solids content. This slurry is then fed by means of high-head diaphragm pumps to a coil-type gangue separator. Live steam is injected into the gangue separator which is maintained under a pressure of 60 psig, heating the slurry to 250° to 275°F and melting the sulphur. The high surface tension of the molten sulphur prevents it from wetting the gangue particles, and collisions caused by eddy currents due to turbulent flow coalesce the small sulphur droplets, which agglomerate into globules up to 2 mm diam. Before leaving the gangue separator, the slurry is quenched to about 190°F by the injection of cold water. Slurry leaves the gangue separator through a pinch-type pressure let-down valve and passes on to a 20-mesh vibrating screen. The agglomerated particles containing from 96 to 98 pct sulphur are retained on the screen and go directly to a sulphur melting pit. Usually about one-third of the sulphur is recovered in the form of these oversize particles, Fig. 2.

The balance of the slurry containing all gangue and non-agglomerated sulphur passes to a conditioning tank. In this tank flotation chemicals are added and pulp density is adjusted to about 20 pct solids. In most instances pine oil, fuel oil, or Aerofloat flotation reagents are used. The quantities required correspond to usual flotation practice, i.e., about 0.10 lb of frother and 0.05 to 0.10 lb of collector per ton of ore. From the conditioning tank

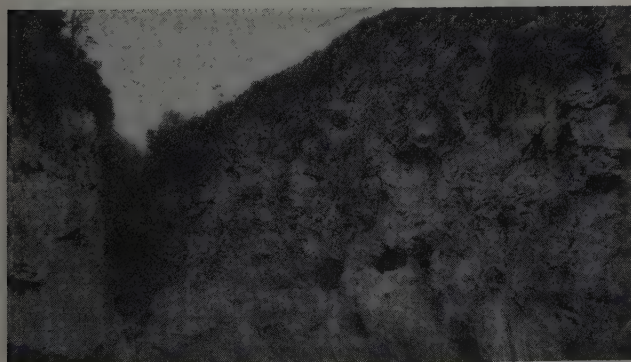


Fig. 5 (left)—Sulphur deposits at 12,000-ft altitude on the slope of Purace volcano, Colombia, S. A. Ore from this deposit is now being treated in the first full-scale plant to utilize the Chemico sulphur recovery process. Fig 6 (right)—Closeup of one of the workings supplying ore to the treatment plant.



Fig. 7 (left)—View of the commercial scale installation in Colombia, S. A. Mill has rubber lining that is further protected by silica blocks.

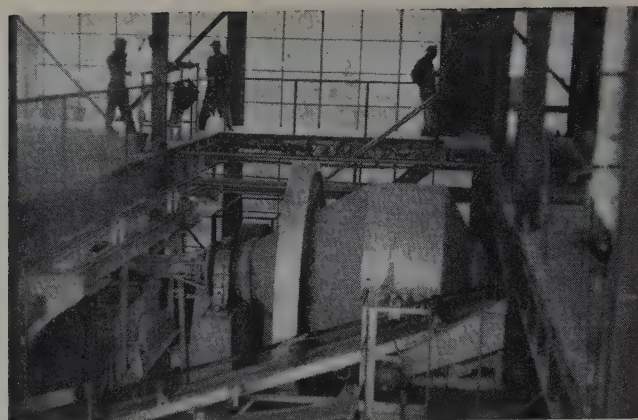


Fig. 8 (right)—Pebble mill in the Colombia plant.

slurry is pumped to a bank of rougher flotation cells which produce a concentrate containing 70 pct sulphur. Tailing from the rougher cells, Fig. 3, containing 2 to 3 pct sulphur, is sent to a thickener which conserves both water and heat, since the clarified hot water is recycled to the process. Thickened tailing is discarded.

The product floated from the roughers passes to a bank of cleaner cells which produce a concentrate, Fig. 4, containing 90 to 95 pct sulphur. The middlings are recycled to the rougher cells through middling thickener which is required to maintain a water balance in the system. Concentrates are dewatered by a horizontal rotary vacuum filter and fed to the same sulphur melting pit into which the oversize sulphur was passed from the screen following the gangue separator. In this pit the sulphur is melted by means of steam coils and the molten sulphur is kept under continuous agitation to maintain impurities in suspension.

The molten sulphur is pumped to a pressure-type filter which yields more than 90 pct of the sulphur originally contained in the ore in the form of 99.5 pct pure sulphur as the final product of the process. The filter cake, which usually contains 40 to 50 pct sulphur, is recycled to the jaw crusher at the beginning of the process.

Engineering Problems

One of the principal engineering problems in developing this process was the selection of suitable construction materials. When sulphur ores of volcanic origin containing silica as the major gangue constituent are ground in water, the resultant slurry has a pH of about 2. Acidity is accounted for by oxidation of some of the sulphur by means of oxygen in the air or possibly oxygen dissolved in the water. The resulting sulphur dioxide forms sulphurous acid, some of which is oxidized to sulphuric acid. While the concentration of sulphuric acid in the slurry does not exceed a few tenths of 1 pct, the slurry is extremely corrosive. Taking into account the fact that ground silica is an extremely abrasive material it becomes evident that the handling of sulphur ore slurries represents a serious corrosion and abrasion hazard. It is known that the failure of several past attempts to recover sulphur from surface ores by means of flotation or autoclaving techniques can be attributed to the lack of corrosion and abrasion-resistant machinery.

Equipment used in the Chemico process is selected with due respect to the above considerations. The pebble mill is lined with rubber which in turn is

protected by a lining of silica blocks. Silica pebbles are used as the grinding medium. The slurry mixing and conditioning tanks are lined with lead and acid-proof brick. For pumping the slurry, rubber-lined diaphragm pumps have been selected, after the testing of a wide variety of alloys in an attempt to find a suitable centrifugal pump. The entire piping, including the gangue separator, is built of No. 316 stainless steel, as are the vibrating screens, the horizontal vacuum filter, and the finishing sulphur filter. Flotation cells are built of California redwood with rubber-covered moving parts. To summarize, the only materials in contact with the hot acidic slurry are No. 316 stainless steel, rubber, lead, wood, and chemical stoneware.

To date more than 50 different ore samples from all over the world have been tested in the Chemico laboratory. The composition of the gangue and the nature of dissemination of the sulphur in the mineral influence considerably the ease of recovery. In general, ores of volcanic origin were found easiest to treat, but with slight modifications in the autoclaving and flotation technique the process was found to be successful on nearly every sample tested. Although some samples proved amenable to beneficiation by flotation alone with reasonable recovery yields, it was noted that samples from neighboring deposits exhibited a remarkable variety of behavior in this respect. After the thermal treatment step, however, samples which were highly re-

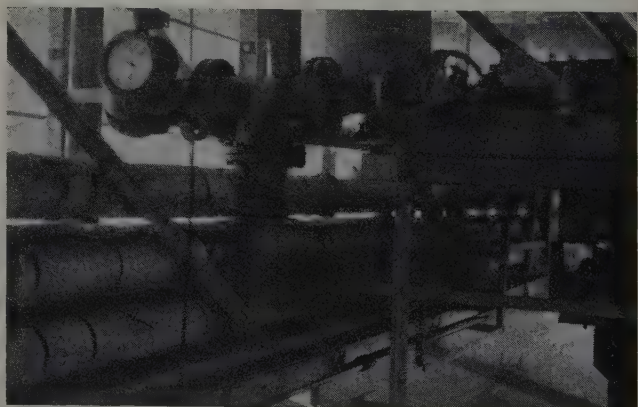


Fig. 9—Closeup of the gangue separator. Here slurry is heated to 250 to 275°F by 60-psig steam, then quenched to 190°F by injection of cold water. Sulphur droplets liberated from the gangue are partly recovered by screening (see Fig. 2), the remainder recovered by froth flotation (see Fig. 3).

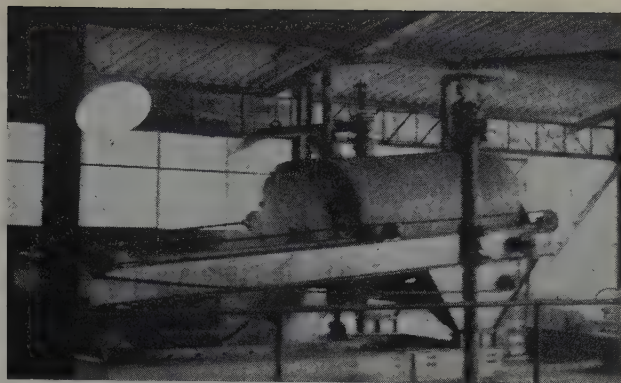


Fig. 10 (left)—Concentrate is dewatered on this horizontal vacuum filter before going to sulphur melting pit. Fig. 11 (right)—Pressure-type filter receives sulphur from melted pit, yields more than 90 pct of original sulphur content of ore in form of 99.5 pct pure final product. Filter cake recycles to crusher at beginning of process.

fractory to ordinary flotation methods yielded very satisfactory results.

Following the work done in the laboratory on small pilot plant scale where quantities of a few pounds were handled, the process was piloted in a plant designed to treat 1000 lb of ore per hr, i.e., 12 tons per day. This pilot plant incorporated all equipment required to carry out the process as outlined above in a continuous fashion. The pilot plant was operated for a 3-month period.

Some pilot plant equipment such as piping and the coil-type gangue separator were considered expendable and were made of ordinary carbon steel. On all these parts severe corrosion was experienced. A steel sand pump used to pump slurry to the flotation cells, for example, corroded and eroded to the point of uselessness within a 24-hr period. All the stainless steel equipment used did not show any signs of corrosion after 3 months operation.

Two principal types of ores both originating from California deposits were tested. Both contained silica as the chief gangue constituent, but they differed in their pyrite and soluble salt, principally FeSO_4 , content, and this difference very markedly affected their behavior in the sulphur recovery treatment.

One ore assayed 33.7 free sulphur, 3.3 pct pyrite, and 6.2 pct soluble salts. This ore was amenable to beneficiation by the standard Chemico technique with a 95 pct sulphur recovery of a 93 pct sulphur concentrate, which after the finishing filter gave a final product of 99.8 pct purity.

The other ore assayed 25.3 pct sulphur, 6.3 pct pyrite, and 11.7 pct soluble salts. The standard flow scheme required certain modifications to obtain satisfactory results with this ore. Since the technique developed in the pilot plant for satisfactory handling of this high pyrite ore was found applicable to other samples containing appreciable amounts of pyrite and sulphate, steps introduced into the process warrant further discussion. The principal modifications consisted of 1—a rougher flotation to double sulphur content of material fed to subsequent parts of the process and filtration and washing of this material to remove the soluble salts and 2—operation of the gangue separator in such a manner that almost no oversize sulphur was produced, so that all sulphur in the ore was recovered as a 90 pct flotation concentrate rather than a combination of oversize and concentrate. This procedure was required since it was found that pyrite particles are wetted by molten sulphur and therefore tend to contaminate the oversize sulphur. Photomicrographs of the interior of the oversize

particles showed interspersed sulphur and pyrite crystals.

After detailed experimental study of the variables affecting agglomeration, a procedure was developed which almost entirely eliminated production of oversize particles in the autoclaving step in the gangue separator. This technique involved the use of an alkaline circuit and a short retention time as well as low temperature in the continuous autoclave. As a result of this investigation, flotation concentrates containing 93 pct sulphur with a 95 pct recovery were successfully produced from the high pyrite ore.

The first commercial installation utilizing the Chemico sulphur recovery process has been constructed in Colombia. This plant is located at the site of a very significant sulphur deposit on the slope of the Purace volcano at an altitude of 12,000 ft, Figs. 5, 6. The plant is designed to produce 33 metric tons of pure sulphur a day from 90 metric tons of ore containing an average of 40 pct sulphur, see Figs. 7-11. Equipment used and significant steps of the process are similar to those described under Process Details, p. 883.

Utility requirements per ton of sulphur produced in a plant of this size and handling a corresponding grade of sulphur ore are tabulated below:

Power, hr	65
Steam at 75 psig, saturated, lb	4450
Cooling water, gal	1900

Three operators and two helpers per shift are sufficient to man a plant up to 1000 tons of ore per day feed.

Detailed investment and operating cost studies for a plant producing 100 tons of sulphur per day, located either in Wyoming or California, show that such an operation would produce sulphur which could sell at a cost competitive with the present controlled price of \$22 per ton.

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The Use of a Caved Block as an Ore Pass and Its Application to Open-Pit Mining

by H. Carroll Weed

By caving a block into the workings of its open pit and using the block as an ore transfer, the Inspiration Consolidated Copper Co. has solved a transportation and sizing problem, making possible a great expansion of open-pit methods as applied to Inspiration orebodies.

FROM 1915 to 1948 the entire production of Inspiration Consolidated Copper Co. was supplied from underground mining. The sole method used was block caving. Underground haulage and hoisting facilities were designed and geared to large-scale production.

Beginning in 1948 open-pit mining was substituted for block caving in a portion of Inspiration's Live Oak orebody. The idea proved so attractive that before the Live Oak pit had come into production, another pit on the Colorado orebody (now the Thornton pit) had been laid out and stripping started. It should be noted that these orebodies were not new and the entire program was one of changed methods of mining.

Since part of these orebodies had already been mined by block caving methods, it must be recognized that haulage levels had been established under the area or adjacent to it. The orebodies lay on the south side of Inspiration Ridge, while the shafts, crushers, and treatment plants are all on the north side of this ridge. The existing main crushing plant was designed to take a maximum of 12-in. material, and all ore was sized to this dimension by passing through grizzlies in the stopes.

In the original planning for open-pit work much thought was given to the transfer of ore to existing underground levels for haulage and hoisting from the regular shafts. If this could be done the necessity of crossing the ridge could be eliminated. Final decision called for cutting a road through the ridge, installation of a primary crusher at an elevation of 3968 ft on the north side of the ridge, and railroad haulage with existing facilities to the main coarse crusher. This decision was brought about by the difficulties of properly sizing and transferring to underground haulage large tonnages of coarse

breaking oxide which lay on the upper benches of the proposed pit. Trucking over the ridge on a 7 pct grade would give a cheaper and more flexible operation in handling this material.

The benches in the pits are laid out at 50-ft intervals and designated according to elevation above sea level. The lowest bench in the original pit was laid out at 3500-ft elevation. All haulage roads are on a 7 pct grade and a vertical lift of 468 ft was considered about the maximum economically possible from a cost standpoint.

After 2 years' operation the advantages of pit mining, both from the standpoint of costs and flexibility of operation, became more and more apparent. Studies were then started to develop the extension of open-pit mining to more of the ore reserve than had been planned originally. The idea of transfer raises was again explored. Obviously, as elevation of the pit was lowered, shorter transfer raises would be needed to reach the main haulage levels. Transportation from the lower elevations to the rim of the pit by belt conveyor or skip hoisting was also considered. However, it was recognized that sizing would be required both for conveying and regulation of feed size to the main crusher plant. This would necessitate a fixed or portable crusher located somewhere in the pit. It was known that the sulphide ore at the lower elevations was softer and easier to break than the overlying oxide of the upper benches. This sulphide ore is very suitable for grizzly sizing.

Calculations indicated that for ore mined on 3650 elevation, costs for trucking to the primary crusher, crushing, and delivery by railroad to the coarse crusher would equal costs for dropping the same material to the 600 level, hauling underground, and hoisting directly to the coarse crusher bins, provided a suitable method of sizing and transferring could be developed. Above 3650 elevation costs would favor surface haulage; below they would favor underground haulage.

The idea of caving an underground block from the 600 haulage level directly into the bottom of the pit

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Discussion on this paper, TP 3614B, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, Oct. 3, 1952. Revised July 9, 1953.

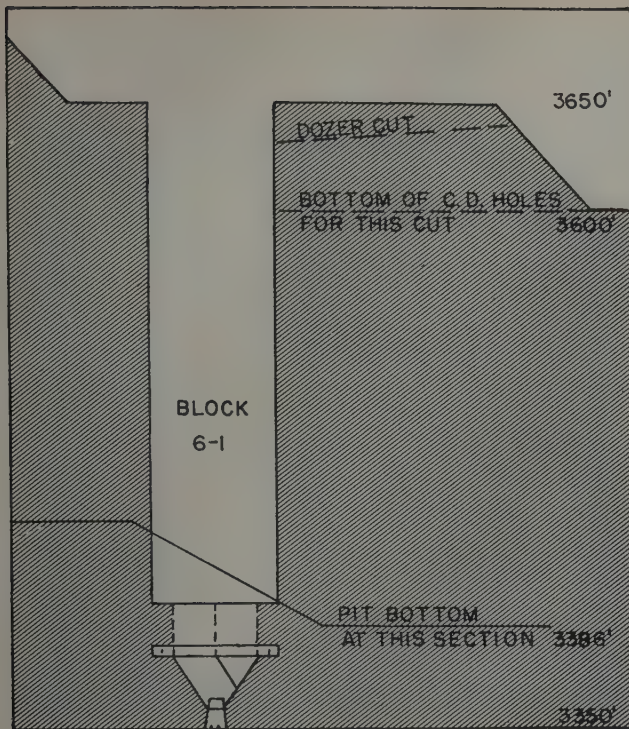


Fig. 1—Section through Block 6-1 and open pit.

and the continued use of this caved block as an ore transfer system developed from the studies made. Such a method provides for sizing, since all ore drawn must pass through the block grizzlies. The block provides plenty of storage capacity and a high rate of draw can be maintained, far beyond that obtainable from a regular transfer raise. Expensive upkeep of a timbered transfer raise is eliminated.

A suitable location was found in the Thornton pit, where the top of the ore was at approximately 3650 ft and where the 600 haulage level, elevation 3350, was reasonably close to the proposed block. Waste was stripped down to this elevation, and at the same time the block was developed on the 600 level. The block was in a granite porphyry which was somewhat altered. Such ground would stand well when timbered and cave freely when undercut, see Fig. 1.

In area the block is slightly smaller than normal, having but three grizzly lines of 33 1/3-ft spacing, three grizzlies in each line spaced on 20-ft centers and a total area of 60x100 ft. The raise system below the grizzlies consists of a single main raise from one side of the haulage drift and a main raise with a branch raise from the other side of the drift. Each of the six chutes in the haulage drift is equipped with an air-operated undercutting type of gate, Fig. 2, replacing the ordinary hand-operated arc type. With these gates, speed and ease of loading have been greatly increased and spillage held to a minimum.

Since it was anticipated that the block would be required to pass a very large tonnage, steel control sets above the grizzlies were used instead of timber sets, and experimental grizzly sets, some of steel and some of wood, were installed.

The block was undercut in the normal manner, but at 23 ft above the grizzly instead of the usual 18 ft. This undercut is at an elevation of 3409 ft above sea level, or 241 ft below the 3650 bench where the top of the ore was exposed. The block was undercut before stripping had been completed and was drawn

very slowly. When less than 10 pct of the estimated tons in place had been drawn the stripping had been completed and the rate of draw was increased. The first hole appeared on surface when the slope was 18 pct drawn. This hole was within the projected block limits and covered about half the slope. The slope had been drawn down about 75 ft when it was observed that it had caved approximately full size to within 75 ft of surface, at which point a back was encountered which resulted in an overhang 50 ft wide on the north side. At this time the open slope was filled by surface mining and has been kept approximately full ever since. When surface mining dropped to an elevation below 3650, it was not possible to keep the hole full below this overhang, whereupon the overhang caved and the surface hole assumed the shape of the upward projected limits of the original block.

When the slope completely caved, in reality it became an oversized ore pass 60x100 ft in area and up to 241 ft in height.

Costs to date have checked with the estimate that 3650 elevation is the break-even spot between overland and underground haulage.

With this ore pass, so-called, in successful operation it has been possible to extend pit plans to include much of the ore reserve lying below the limits of economical trucking operations. Several locations

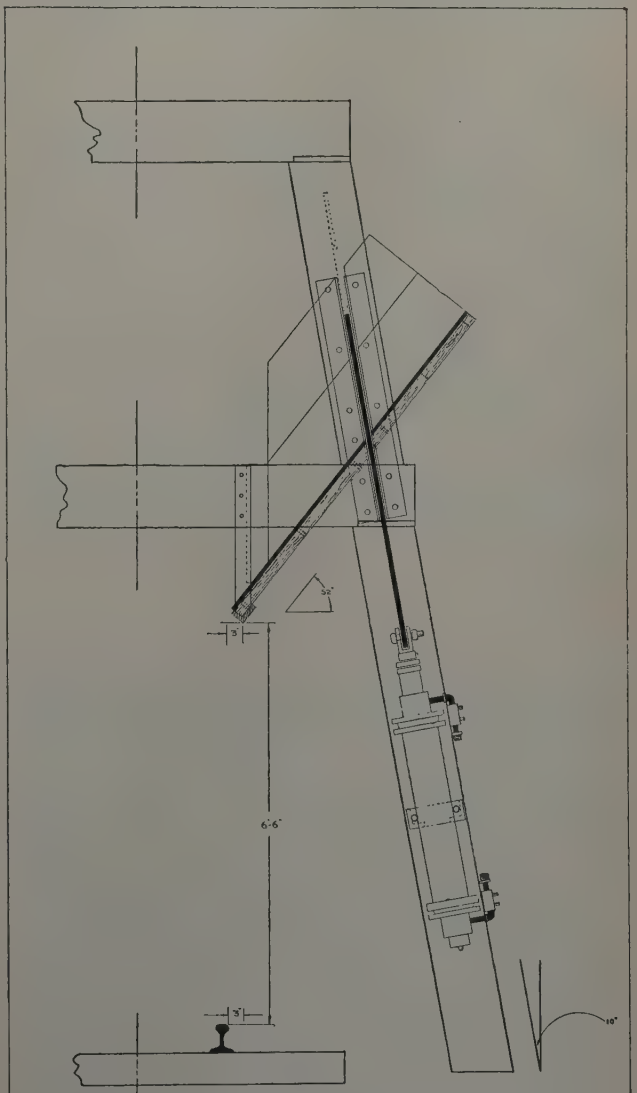


Fig. 2—Assembly drawing of guillotine-type chute gate installation.



Fig. 3—Dozers and carryalls working in cut below 3650. Transfer (dotted outline) full and not being drawn. Loads dumped on top of transfer. Shovel in background stripping waste on 3700 bench.



Fig. 4—Transfer being drawn. Loads dumped at edge to be pushed in by dozers. Cut below 3650 is shown.



Fig. 5—A load approaching side of the transfer for dumping while the transfer is active.

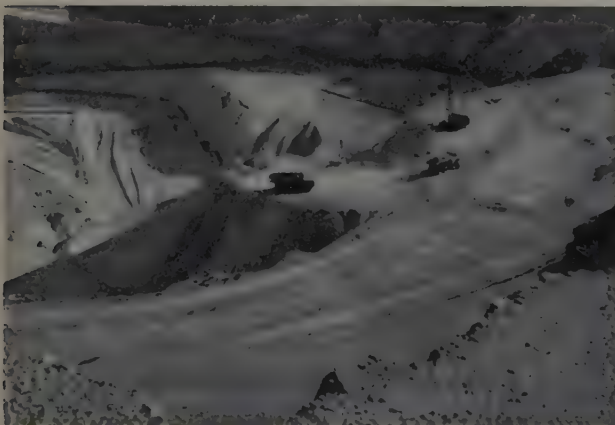


Fig. 6—Trucks dumping into transfer from 3700 bench. The 3650 bench is shown in background and the cut made by dozers and carryalls below 3650 at left center.

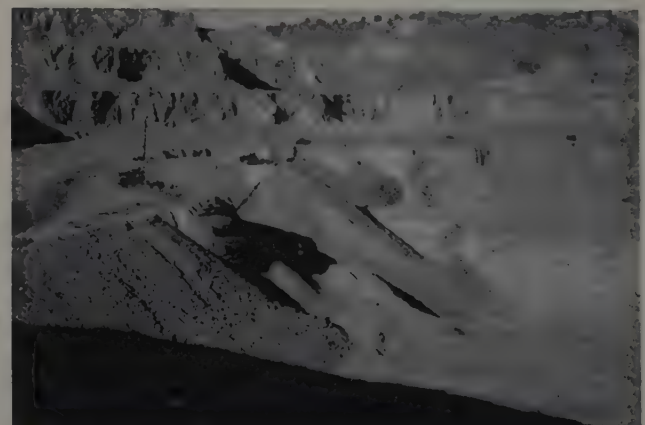


Fig. 7—A truck is shown dumping into the transfer from the 3700 bench. The 3700 bench and the scraper cut below 3650 appear at the right.

for similar blocks will be available as the Thornton pit progresses. As the 600 level is passed in the downward mining of Thornton pit, the 800 level will become the haulage level. Similar caved block ore passes are planned for the Live Oak area, where the 600 haulage level will serve for the life of the pit.

Mining Methods Used With Caved Block Transfer

Glory hole mining is an old method. Ore transfers, of limited cross section, to transfer surface-mined ore to underground are not new, the method having been used by United Verde in Jerome, by Phelps Dodge at the Sacramento pit in Bisbee, and probably by others. Through necessity Bagdad engineers used a caved block as an ore pass when developing their open pit. But so far as is known, no other company has deliberately caved a block into an open pit for the purpose of using it as an ore pass.

Since the opening is an ore pass it is desirable to maintain the walls as nearly vertically above the stope as possible. The mining method does not correspond to glory holing, as the hole is not progressively enlarged and emptied until the angle of repose is reached. Beyond a distance of approximately 1000 ft standard open-pit mining will be used, with trucks dumping into the transfer. This has been tried and found satisfactory.

Experience indicates that ore within 1000 ft of the transfer can be handled by bulldozers and carryalls, Fig. 3, at lower cost than by shovels and trucks. With this method, regular 50-ft churn drill holes are drilled and blasted in a 20-ft center pattern. Multiple lines are blasted and a channel is cut, with caterpillars and carryalls, sloping toward the transfer. The sides of this cut are kept on a 45° angle. When the cut has reached a depth of 50 ft, or less if desired, churn drill holes along the sides are blasted toward the cut in single or multiple rows and the ore cleared out again to a 45° angle. This process is repeated until the economical limits of the caterpillar and carryall are reached. Shovels and trucks take over at this point.

Drawing in a transfer of this area produces almost immediate subsidence on surface. There is little or no chance of a hang-up with its subsequent abrupt subsidence. Actually, the carryalls can run out over the full transfer and dump their loads while the ore is being drawn from below. For safety's sake, however, no equipment is allowed to cross the transfer while it is being drawn. Present practice is to draw the ore on one shift, Fig. 4, and refill the transfer on another shift, Fig. 5-7. When it is necessary to work on surface on the same shift in which ore is being drawn below, the carryalls dump their loads at the edge of the transfer and the ore is then pushed in with a bulldozer.

Four D-8's and two Model 90 caterpillar carryalls are the normal working equipment on surface. Five thousand tons per 8-hr shift can normally be transported by this equipment. Spare equipment consists of two smaller carryalls. Three other D-8's, normally assigned to other work, back up the motor power as spare units. In addition to lowering costs, use of this type of equipment for all ore within 1000 ft has released shovel and trucks to necessary stripping.

All truck haulage to the transfer will be a level haul. This should cut down on truck maintenance, and it is obvious that fewer trucks will be needed to transport any given tonnage. As the benches are lowered the transfer will be cut down 50 ft at a time, and it is quite probable that one side of the

transfer may be 50 ft higher than the other. This will allow ore haulage from two benches.

In a test run with the shovel located 400 ft from the block, 6498 tons were truck-hauled into the block in a single shift. Only two trucks were used. Conditions were ideal and continuous production at this rate could not be expected.

It has been found that the ore breaks very well and little sizing is necessary on the grizzly level. Undoubtedly there is some crushing action in passing downward 240 ft. The normal crew for drawing and hauling consists of 3 men in the stope with from 8 to 10 men on the haulage crews, depending on the number of trains used. A 3-train operation is probably the most efficient. With 24 4½-ton car trains and 3 trains in operation on the 2500-ft haul to the shaft, as much as 6150 tons have been drawn in an 8-hr shift. A 5200-ton average per shift under these conditions has been maintained over a period of 2 weeks.

For the six months ending July 1, 1953, an average of 124 tons of ore per manshift was obtained in the combined operation. In the open pit this includes all shifts spent on drilling and blasting and transportation to the block, part by cabs and carryalls and part by shovel loading and trucks. Underground shifts include chute tapping, hauling and hoisting, and such repairs as were needed. Also included are miscellaneous shifts prorated to various mining operations on a shift basis, such as supervising, engineering, timekeeping, and ventilation. Two hoistmen are included. The hoistmen and most of the miscellaneous shifts, although their services are charged to the operations, would be necessary even if no ore were hoisted.

The draw from the transfer is not always regular, as this particular ore is used as a balance in obtaining both the desired grade and oxide-sulphide ratio of plant feed.

Repairs on the grizzly level and in the drawing area have been light. One timberman part time has been able to keep up with the necessary work.

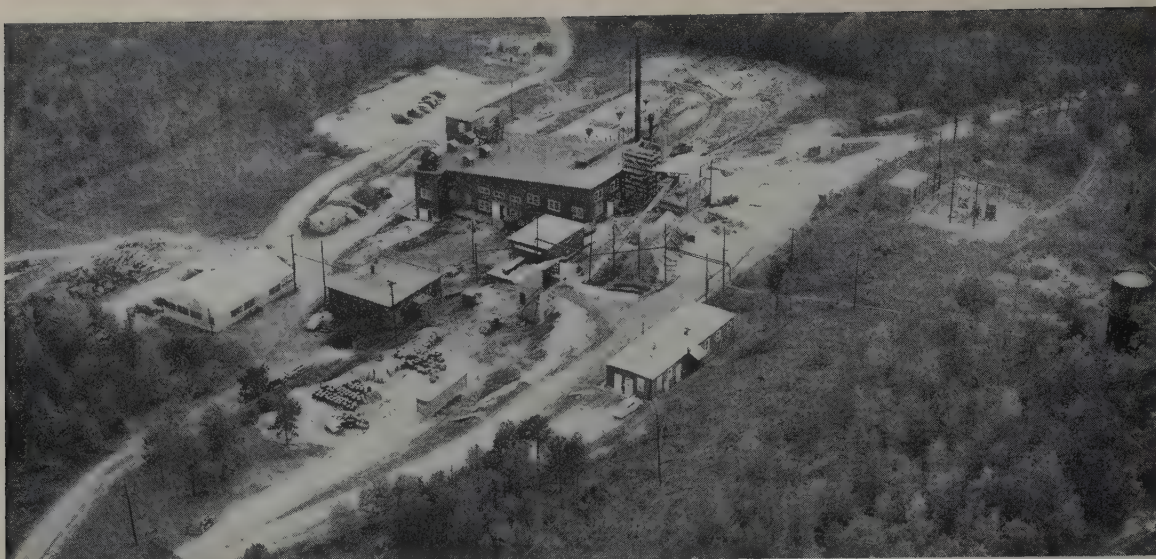
It has been found that the steel grizzly sets are not much better than timber sets, but steel-control sets are almost essential. The air-operated doors on the haulage level are especially satisfactory. With 3 pairs of chutes it is seldom that ore is not available at each chute position so that 3 cars can be loaded simultaneously. With the development of subsequent transfer blocks undoubtedly improvements will be incorporated, based on experience gained with this initial block. Such changes will be mainly in re-enforcing points of wear and streamlining the flow of ore. This first transfer block up to July 1, 1953, has passed about 1¼ million tons of ore.

Summary

The use of a caved block for a transfer raise at Inspiration has 1—solved a sizing and transportation problem, allowing a sizeable increase in tonnage that will be available for mining by open-pit methods; 2—decreased costs on all ore originally planned for open pit lying below the 3650 elevation; 3—solved the sizing problem without installation of a crusher; 4—made use of existing facilities and equipment which were idle, and 5—released shovels and trucks for other work.

Acknowledgment

The writer wishes to thank members of the operating staff at Inspiration for their help in preparing this paper.



Mining and Milling of Lithium Pegmatites At Kings Mountain, N. C.

by E. R. Goter, W. R. Hudspeth, and D. L. Rainey

THE area in which spodumene-bearing pegmatites occur extends from Gaffney, S. C., in a northerly direction to Lincolnton, N. C., a distance of about 16 miles. The zone averages 2 miles in width.

Interest in this area was first aroused by the discovery of small amounts of cassiterite in the pegmatites and in greissen which occurs on the walls of some of the pegmatites. Attempts were made from 1880 into the 1920's to mine and concentrate cassiterite, but there were no successful operations. In 1935 L. M. Williams became interested in the area and began prospecting by trenching and sinking several small shafts. He gradually gained control of a considerable acreage south of the city of Kings Mountain, N. C. Mr. Williams states that he produced some ore from a shaft which he shipped to the Maywood Chemical Co. in the late 1930's. In 1937 G. H. Chambers of the Foote Mineral Co. visited the area and examined several properties, but the lithium market was too small to support the necessary concentrating plant.

Early in World War II the Solvay Process Co., encouraged by various federal agencies, acquired the Williams properties and leases and started a drilling program to develop reserves for spodumene mining. A study of this area was also made under T. L. Kesler by the U. S. Department of the Interior Geological Survey.

Starting in 1942 a concentrating plant, water supply dams, and shops and other buildings were constructed. The Superior Stone Co., a neighboring crushed stone operator, was awarded a contract to

mine and crush the pegmatite ore. Late in 1943 the plant started operations, which were continued until February 1946. A drop in demand for lithium chemicals forced it to close and remain closed for several years.

In 1948 the Foote Mineral Co. became interested in assuring itself a source of lithium ore on the North American continent, to be used in the production of lithium ore chemicals, and inspected properties in Canada, South Dakota, and North Carolina. The North Carolina deposits were the most promising. In the fall of 1950 the company leased the Solvay interests with option to buy and acquired additional properties. Renovation of the mill and preparation of the quarry for mining was begun in earnest in January 1951.

A new crushing plant was built and construction started in July 1951. The flotation process used was developed in the North Carolina State Minerals Research Laboratory located at Asheville, N. C.

Foote is now in the process of expanding its production. The crushing plant is being revamped and much of the old milling equipment purchased from Solvay will be replaced by new. Mining equipment of suitable capacity has been received and is in use. The frontispiece is a general view of the mill.

The orebodies consist of pegmatite dikes composed of fine-grained albite and quartz, with varying amounts of mica and spodumene. The latter may amount to as much as 50 pct of the rock, but the overall average is about 15 to 20 pct. Tin, columbite-tantalite, beryl, and other valuable minerals are present in varying small amounts.

These dikes, which intruded into the Roan Gneiss underlying most of the Piedmont area, strike approximately N 30 E and dip about 85° to the west. The gneiss has the same strike but dips about 70° west. The dikes lie in a belt extending from Lincolnton, N. C., to Gaffney, S. C., and passing through

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Discussion on this paper, TP 3636H, may be sent (2 copies) to AIME, before Nov. 30, 1953. Manuscript, Dec. 30, 1952. Revised June 23, 1953. Los Angeles Meeting, February 1953.

the town of Kings Mountain. For years this was known as the Carolina tin belt and was worked in numerous places for cassiterite. The tin content was spotty, however, and no large-scale production was ever achieved.

The first published account of tin in the belt appeared in 1884,¹ but from that time until as late as 1931 the existence of spodumene is scarcely mentioned except as a constituent of the tin-bearing pegmatites. By 1942 one small quarry was opened for the mining of this lithium ore.² According to Mr. Kesler this pit was 100 ft long, 12 ft wide, and 35 ft deep.

During the period from 1942 to 1944 the Solvay Process Co. which operated this property dug eight trenches for a total of 1600 ft to determine the extent and depth of the orebodies. One trench was about 650 ft long and crossed three sizable dikes. Two churn drill test holes were put down for a total depth of 270 ft. Seventeen inclined diamond drill holes were drilled, a total footage of 10,011 ft. The deepest of these was stopped when it had reached a vertical depth of about 750 ft.

Since beginning operation in 1951 Foote Mineral Co. has carried on a continual program of prospecting. Considerable detailed geologic mapping has been done, necessitating 2000 ft of trenching by bulldozer. A program of diamond drilling is being conducted to supplement information obtained from core holes drilled by Solvay Process Co. and to outline more accurately the orebodies at depth.

Past and Present Mining

The quarry in existence in 1942 was on the east side of pegmatite No. 25 which was 85 ft wide at the top and believed to be at least 1450 ft long. This became the starting point for Solvay Process Co. operations. They increased the pit to about 400 ft long, a maximum of 60 ft deep, and the full 85-ft width of the dike. Mining at this time consisted of drilling the rock in 20-ft benches with wagon drills. After the large rocks were blasted and blockholed, the ore was loaded by power shovel into trucks and hauled to the crushing plant. Benches were carried across the width of the dike and the face was advanced along the strike.

Solvay also began mining the north end of pegmatite No. 21 at a starting point 60 ft wide, advancing the quarry as a side-hill operation by going in from one end and carrying the bottom level. At an advance of 300 ft the quarry had obtained a maximum depth of 40 ft and was being worked from two benches. This pit was abandoned because the ore was weathered and an acceptable grade of concentrate could not be made at that time.

Foote Mineral Co. began operations by dewatering the quarry in pegmatite No. 25 and continued with much the same procedure Solvay had used. By October 1952 the pit had obtained a depth of 120 ft and was over 700 ft long. Including stripping to obtain safe walls, it measured about 150 ft wide at the top. The ore had narrowed to less than 50 ft wide at the bottom. Since the ore pinched in very rapidly in the south end, and a horse of waste in the north end made advancement in that direction uneconomical, working length was limited. The quarry in the north end of pegmatite No. 21 was also cleaned out, but it was found that this ore still could not be successfully milled.

To open a new quarry, the top, end, and sides of pegmatites No. 18 and 21 were stripped at the south

end for a distance of about 300 ft. Stripping bore out indications that these two dikes were very near together. To obtain a longer working face and therefore a more flexible operation, it was decided to mine across the strike of the orebody. Accordingly a face approximately 200 ft long was opened along the footwall side where the country-rock had been stripped back for a distance of 50 ft to a depth of 20 to 30 ft. However, because of the weathered condition of the ore and the presence of much clay in the seams, a large portion of this ore had to be discarded as waste. Even then the mill had difficulty making a good product.

An additional quarry was opened on the largest of the pegmatite bodies in the area lying some 2500 ft north of pegmatite No. 25. This dike, known as pegmatite No. 8, has a maximum width of 300 ft and is well over 1000 ft long. Because of the topography it was possible to develop a face 200 ft long by clearing the trees and making a road along the footwall. At the present time mining is being carried on simultaneously with stripping operations. The top 6 to 8 ft is being wasted, as it contains most of the dirt and clay so detrimental to the milling operations. The shovel operator loading ore for the crusher, Fig. 1, sorts it to remove as much of the waste as possible. This material is then dumped over a scalping grizzly to remove any remaining dirt. The resulting ore, while stained, is free of the clay and can be successfully concentrated. The first bench in the pegmatite No. 8 quarry started out at 16 ft, but because of the rising ground it will reach a height of about 30 ft. Subsequent benches will be kept at 20 ft. This is due to the fact that the fractured nature of the orebody makes the use of higher benches unsafe.

Present equipment in the mine includes air compressors with a combined capacity of about 900 cfm, two 4-in. wagon drills for drilling benches, two jackhammers, and one 3-in. wagon drill used in stripping. A Northwest 25 shovel and a Bucyrus-Erie 38-B shovel are used for both stripping and a loading ore, and a P & H 255A crane with a 40-ft boom and 4000-lb ball is used for secondary breaking. Two 10-ton Euclid trucks are used for hauling, supplemented by two 2-ton GMC trucks hauling on a contract basis. A TD 18 with hydraulic dozer blade is used for clearing, cleaning, and roadwork.

Future development will, of course, depend upon the character of the dikes being worked. There are no plans at present for any underground operations, since there is at least 10 years' reserve of ore for surface mining. Orebodies in future will be developed by stripping the eastern footwall to give a 30-ft



Fig. 1—Loading rock in open pit for trip to jaw crusher.

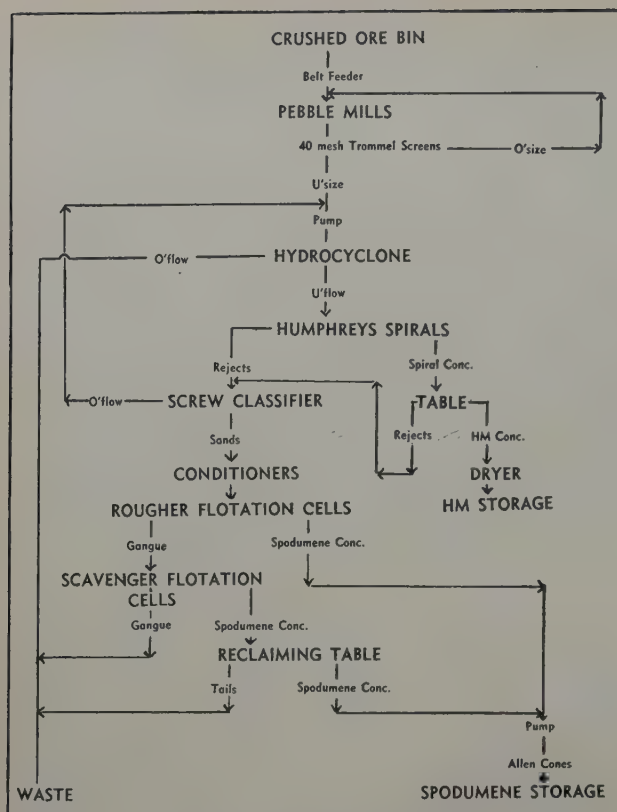


Fig. 2—Mill flowsheet.

roadway a slope of about 1 to 1 in the waste. The face will then be carried across the strike of the dike and the hanging wall sloped at whatever angle is necessary for safety. This will give a safer operation than the method used in pegmatite No. 25 and also make it possible to recover some ore from small dikes which lie close to the larger orebodies.

Ore from the mine is dumped onto a 10-ft by 36-in. Telsmith apron feeder, which feeds an 18x36-in. Traylor jaw crusher. The $\frac{1}{2}$ -in. jaw crusher product is delivered by conveyor to a 4x10-ft Ty-Rock screen clothed with $\frac{3}{4}$ -in. Ty-Rod screen cloth. Screen undersize is delivered by chute to a bucket elevator which discharges into the crushed ore bin. Oversize from the screen is fed to a 436 Allis-Chalmers Hydrocone crusher, set to discharge a product about 66 pct $\frac{3}{4}$ in., which is returned to the primary conveyor feeding the Ty-Rock screen. A mill flowsheet is shown in Fig. 2.

A belt feeder equipped with a variable speed drive delivers the crushed ore at a rate of about 360 tons per day to two 6x8 Hardinge pebble mills, lined with 30-lb rail and charged with 3x4-in. granite blocks as grinding media. The mills are driven with 125-hp induction motors, and grinding charge is maintained by adding media until the motors are pulling 150 amp at peak of surge. The mills are equipped with trommels clothed with nominal 40-mesh Ton-Cap screen. About 20 pct of head feed reports as screen oversize in the mill discharge and is returned to the heads of the mills by an Esperanza drag. Screen undersize is laundered to the pump-box of a 4-in. Wilfley pump.

The screen undersize is pumped at about 15 pct solids to a 12-in. hydrocyclone for desliming. The hydrocyclone is centered over a round gathering box with eight wedge-shaped compartments which empty into the feed boxes of eight 5-turn, 24-A Humphreys spirals. The cone-shaped spray dis-

charge of the hydrocyclone effectively distributes underflow to the spirals. About 5 pct by weight of the spiral feed is removed from the ports as a rougher heavy mineral concentrate and delivered to a table for further concentration. The spirals rejects are laundered to a 21-ft x 48-in. Wemco spiral classifier, which discharges the sands into No. 1 conditioner.

Flotation

Two conditioners in series are operated at about 55 pct solids. The following reagents are added at this point: amine, a silicate gangue mineral collector; caustic, a pH regulator; dextrine, a selective depressant for spodumene in an alkaline circuit; and pine oil as a frother. It has been found that a delicate balance exists between the amine and the pH in the conditioners. Dextrine depresses most effectively in a highly alkaline circuit; the collecting properties of the amine, however, are sharply reduced when the pH of the pulp goes above 11.3. The conditioner discharge is diluted to about 25 pct solids and delivered to the feed box of a bank of six No. 24 Denver flotation cells. In this separation the float product contains the gangue minerals: quartz, mica, feldspar, and some hornblende. The underflow is the spodumene concentrate which is pumped to an Allen cone, dewatered, and delivered to storage. The rougher gangue froth containing about 7 pct spodumene is delivered to a bank of six No. 22 Morse-Weinig cells for scavenging. The scavenger machine discharge, usually not of concentrate grade, is pumped to a Deister table for gravity concentration. The table tails are pumped to waste and the table concentrate is combined with the rougher cell spodumene concentrates.

For mill control purposes, the spodumene content of any given sample is determined as follows. A weighed sample is placed in a centrifuge tube containing acetylene tetrabromide, sp gr 2.96. The tube is spun and the spodumene, with a specific gravity of about 3.2, sinks in the liquid. The floats are removed and the sinks are filtered, washed with benzene, dried, and weighed. The percent sinks is roughly equivalent to the percent spodumene. An error is introduced, however, if any quantity of hornblende is present.

The concentrates drain to about 95 pct solids on the storage pads, see Fig. 3. A scoopmobile transports the concentrates to a hopper from which the material is fed to a Ruggles-Coles rotary drier. Spodumene concentrates to be used in the manufacture of ceramics are passed over an induced-roll



Fig. 3—Spodumene on draining pads awaiting drying and bagging operation.

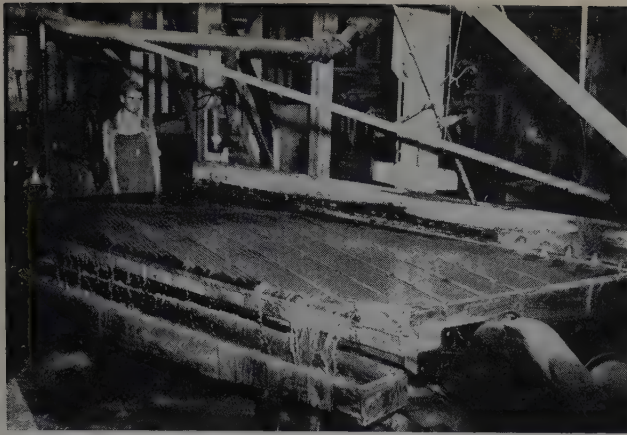


Fig. 4—Shaker table for recovery of tin columbite by-products of spodumene process.

type magnetic separator, where iron and iron-bearing minerals are removed. When the iron content is not critical the concentrates are loaded and shipped immediately after drying.

At present a heavy mineral concentrate is the only by-product being made. The rougher spiral concentrate mentioned before is delivered to a Deister table, Fig. 4. The table concentrate consists of cassiterite, columbite, and tantalite, with small quantities of pyrrhotite, pyrite, and monazite, and is produced at the rate of approximately 80 lb per day. Table rejects are pumped into the spiral classifier and returned to the flotation circuit.

Feldspar concentrates have been made in the plant by use of amine, sodium fluoride, and sulphuric acid, but the iron content in the concentrates has been over a commercial limit because of the presence of hornblende. The iron-bearing minerals can be removed by magnetic separation. Another deterrent to plant-scale production of feldspar is that with the relatively fine grind necessary for the beneficiation of spodumene the feldspar is very fine and consumer prejudice against a product of this nature will have to be overcome.

The mica content in the head feed varies from almost nothing to about 2 pct. For a time a mica concentrate was produced by removal of the slimes portion from the spirals used for heavy mineral production. Several tons of a commercially desirable mica concentrate were produced per month by this method. However, the operation of a spiral for mica production and for heavy mineral concentration is not the same. It was found that heavy mineral production was impaired and the production of mica was temporarily abandoned.

The Foote Mineral Co. is cooperating with the Bureau of Mines in an intensive investigation of recovery of beryl from the pegmatites. This work shows some promise and will be tried on a plant scale as soon as conditions permit. The beryl in the pegmatites is of the clear variety and is very difficult to identify.

Construction has begun on an expansion program which will raise the capacity of the operation from the present 360 tons per day to 1000 to 1200 tons per day. Changes to be made in the crushing plant include the installation of a 26-in. Traylor primary gyratory crusher and the conversion of the present 436 Allis-Chalmers Hydrocone crusher to a 636 crusher for the production of 1¼-in. material.

Two 5x12-ft Marcy rod mills will receive ore delivered by two Hardinge constant weight feeders.

The discharge of the rod mills will be delivered to 54-in. Wemco spiral classifiers, which will make a separation at approximately 40 mesh, the screw sands returning to the mills. Classifier overflow will be deslimed by hydrocyclones and the sands passed over 32 Humphrey spiral classifiers for recovery of tin and columbite. A Wemco classifier identical with the one presently in use will receive the spiral rejects from one set of spirals. Two banks of No. 24 Denver Sub-A Cells will receive the conditioner discharge and present equipment will be placed in service for scavenging the rougher gangue froth products.

The very desirable feature of the expanded milling facilities will be the two parallel circuits, each capable of handling 500 to 600 tons per day. This will allow proper scheduling of maintenance and partial production in event of equipment failure.

Power, which is furnished by the Duke Power Co. at 44,000 v, is reduced to 440 v for distribution to the plant. Where the power lines to the water-pumping stations and compressors are long, the voltage is again stepped up to 2300 v. Newer equipment is being installed with 2300-v motors.

Tailings disposal is conventional, with an earth tailings dam and Weir-type overflow into culverts buried in the dam. The overflow Weirs are 24-in. diam concrete pipe, the first one having been set in a concrete housing which conducts the overflow into the discharge culverts. The Weir is raised by addition of 1-ft sections of concrete pipe as needed.

Service Departments

Repair, maintenance, and construction are organized as a separate department, headed by a plant engineer and directly supervised by a maintenance supervisor and two foremen. Except for the maintenance supervisor, who had been an electrician for Solvay Process Co., the majority of the personnel in this department had no experience in the mineral production field and were trained after being hired. It has been necessary in the past to contract major construction jobs, such as the building of the crushing plant. It is planned that in the future much of this can be taken care of by the maintenance department. The plant was idle for 6 years and had been furnished largely with second-hand equipment. The 24-hr 7-day operation in the mill placed a heavy burden on this equipment, with resulting high maintenance charges.

An employee welfare supervisor heads the safety committee, which holds monthly meetings and makes regular inspections. Accidents that have occurred are discussed and future prevention outlined. Protective clothing is furnished when needed.

All shipping is done from a loading station located on the main line of the Southern Railroad about 1½ miles from the plant. Here a hopper-conveyor belt-bucket elevator system is used to load bulk shipments into hopper cars. It is also used together with a car loader to load box cars. A loading platform is utilized in loading other shipments. Both bulk and bagged products are hauled to the loading station by truck.

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Economics of Pegmatites

by Paul M. Tyler

MUCH information concerning pegmatites which was thought to be true a few years ago has been proved false, and what is now actually known about some pegmatites is not true of many others. The erratic and seemingly unpredictable structure and variable composition of this class of mineral deposits has been widely emphasized. Even parts of the same pegmatite body exhibit marked differences in texture, mineral composition, width, and attitude. Constructive geological thinking in respect to pegmatites now aims to establish general laws rather than to stress the confusing diversity of features having no special economic significance.

Substantial progress has been made in classifying different types of these deposits according to general features, internal structure, mineralogy, and origin. In some cases it has even been possible to block out tonnage reserves in advance of mining. It is still easy, however, to make highly erroneous predictions after a preliminary examination of a pegmatite prospect.

Pegmatites are important to the economic well being of the country and to its military security. They furnish virtually all the feldspar, strategic mica, beryl, columbium, tantalum, and caesium utilized in the United States, as well as sundry other minerals and significant amounts of lithium and rare earth minerals and gems. With the exception of vermiculite, occasional ilmenite-rutile, and perhaps soda-lime feldspar and garnet deposits, basic pegmatites are of little economic importance. Consequently in this paper, as in common parlance, the term *pegmatite* generally relates to coarse-grained acidic rocks or what is aptly called *giant granite*.

Available data indicating the size and importance of the production and trade in specified pegmatite minerals are summarized in Table I.

Geological Features

Much of the latest thinking on the economic geology of pegmatites is now available in a 115-page monograph¹ by a group of experts who participated with geologists of the Federal Geological Survey in the widespread wartime investigations. Doubtless the most significant feature of the monograph is indicated by the title, *The Internal Structure of Pegmatites*, but it also contains a vast amount of other new information and includes the assimilated concepts of many earlier writers, whose works are given in a comprehensive list of references.

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Discussion on this paper, TP 3613K, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, Nov. 21, 1952.

The shape, size, attitude, and continuity of many pegmatite bodies is controlled by the structure of the older rocks in which they occur. If the older rocks are easily penetrated, e.g., biotite schist, most of the pegmatites in a given district will be found outside the parent granite mass as exterior pegmatites. Marginal pegmatites are more prevalent if the older rocks are massive, unsheared, and sparingly jointed. Networks of pegmatites are abundant in highly-jointed rocks. In strongly foliated schists the bodies are usually lenticular, whereas in highly-folded areas they assume tear drop, pipe or pod-like, bow-shaped, or sinuous forms. Jahns² recognizes five major shape classes: 1—dikes, sills, pipes, and elongate pods; 2—dikes, sills, pipes, and pods with bends, protuberances, or other irregularities; 3—trough- or scoop-shaped bodies with or without complicating branches; 4—bodies with the form of an inverted trough or scoop; and 5—other bodies, including combinations of the above and miscellaneous shapes.

Many pegmatite deposits are scarcely big enough to be recognizable as such. Most of them, in fact, are small tabular deposits less than 4 in. wide and usually without economic concentrations of minerals. On the other hand, some pegmatites are more than a mile long and over 500 ft wide. The ratios of length to breadth range from 1 : 1 to 1 : 100. Although the vertical dimension bears no invariable relationship to strike length, tabular deposits or large lenses are often symmetrical enough to show nearly as much continuity down dip as in their horizontal extension, and some pipes or pods are amazingly persistent in the vertical plane.

Small pegmatites often string along a fairly definite trend line; in a given district major bodies may lie roughly parallel, and where only a few of them do not, the erratically disposed bodies generally differ in composition from those conforming to the regular pattern. This does not apply, however, in all districts.

Characteristically, pegmatite veins pinch and swell or split into branches. When they pinch out entirely it is often possible to find a new body by prospecting the extension of the strike or dip, but the chances of finding a hidden deposit are ordinarily too uncertain to justify much subsurface prospecting. Diamond drilling may yield valuable information as to the continuity of known deposits whose upper portions are well-exposed. Some deposits, in fact, can be proved up for hundreds of feet by surface trenching and then intersected by drill holes at various depths like any other vein-like deposit. Others twist and branch, apparently defying all efforts to explore them short of actual mining.

The internal lithologic and structural units that can be recognized in many pegmatite bodies have been listed by Jahns² as follows: 1—fracture fillings, generally tabular bodies that fill fractures in previously consolidated pegmatites; 2—replacement bodies, formed at the expense of pre-existing pegmatites with or without obvious structural control; and 3—zones or more or less complete shells whose boundaries may be roughly parallel to the outer walls of the main body. These three types of units are distinguished by differences in mineralogy or texture or both. The contacts between units may be knife-edge sharp or diffused over several feet of broad gradations, and they need not be uniform.

Some pegmatites seemingly lack systematic orientation, and detailed mapping and careful observation are required for the observer to recognize an orderly arrangement of structural units. Zoning may be incomplete. It may be absent in parts of the body and develop on only one side or at one end. Instead of concentric shells, zoning may exist only as lenses, layers, or elongated pods.

Perfect zoning would comprise a series of concentric shells surrounding a central zone or core. Heinrich,³ for example, concluded that zones may be formed by crystallization of the pegmatite magma in successive layers inward from the contacts. The coarser texture of inner zones can thus be explained by slower cooling.

The four principal types of zones have descriptive names: border zones, wall zones, intermediate zones, and innermost zones or cores. Any of these, especially intermediate zones, may be absent or virtually undeveloped. On the other hand, some pegmatites contain several intermediate zones.

Fifty or more minerals have been reported as occurring in pegmatites, and specimens of as many as 30 varieties are frequently found in the same body. Studies of the relation between structural features and sequence of mineralogical units have indicated that the mineral assemblages are related to zoning and change in approximately the following order, starting from the wall rock and progressing inward: 1—plagioclase, quartz, and muscovite; 2—plagioclase and quartz; 3—quartz, perthite, and plagioclase with or without muscovite and/or biotite; 4—perthite and quartz; 5—perthite, quartz, plagioclase, amblygonite, and spodumene; 6—plagioclase, quartz, and spodumene; 7—quartz and spodumene; 8—lepidolite, plagioclase, and quartz; 9—quartz and microcline; 10—microcline, plagioclase, lithia mica, and quartz; and 11—quartz.

Few, if any, pegmatites contain all 11 of these zones, but those that are present are likely to be grouped in this order. As indicated by this arrange-

ment, the processes of albitization, muscovitization, and the development of lithium minerals are important stages in the formation of complex pegmatites. Other accessory minerals also show a similar sequence. Beryl, for example, may be more abundant in the wall zone, lean in the intermediate zones, but occasionally present as a single giant crystal or nests of crystals in or near the core. Undoubtedly the small crystals and more or less corroded anhedral grains of beryl associated with plagioclase, mica, and quartz in and near the wall rock were formed first, whereas second-generation beryl is represented by the more familiar occurrence of large anhedral or corroded crystals associated with albite, potash spars, quartz, muscovite, and lithium and/or tantalum minerals in zones parallel to the outer edges of very coarse-grained cores.

In two-zone pegmatites, the cores may comprise medium-to coarse-grained plagioclase, potash spars, quartz, mica, and tourmaline. Some of the most productive mica deposits are composed principally of coarse plagioclase and quartz. In general, however, the present writer believes that pegmatites with relatively simple zonal structure or none at all are poor in commercial minerals other than feldspar and that unless albitization is present the search for other minerals including strategic mica books may be unrewarding.

Useful Minerals in Pegmatites

Feldspar: The bulk of tonnage output of all varieties of pegmatite minerals is feldspar. This is the main constituent of most pegmatites and the feldspar-rich zones are commonly the largest and most persistent. Granitic pegmatites are still the principal sources of commercial production, although other occurrences, including massive aplites and coarse-grained granitoid rocks, are coming into the picture. About 60 pct of the composition of all igneous rocks is made up of feldspars, but petrographic examination or chemical analysis shows that feldspar grains in most unpegmatized rock masses are heavily shot with iron-bearing minerals. Another limitation on the number of deposits that can be utilized as commercial sources is the fact that the spar in many igneous rocks is anorthite, or some other lime-rich refractory variety, rather than albite or the even more generally useful potash spars.

Commercial feldspar deposits commonly form cores in potash-rich bodies. In some mines, however, the potash-rich core is more or less surrounded by plagioclase-rich zones, one or more of which may be a productive mica deposit. Although high-grade deposits of both mica and feldspar are

Table I. United States Production and Imports of Pegmatite Minerals

	Production, Mine Shipments				Imports for Consumption			
	1940 to 1944, Avg		1945 to 1949, Avg		1951		1951	
	Short Tons	Value, \$	Short Tons	Value, \$	Short Tons	Value, \$	Short Tons	Value, \$ *
Feldspar	354,228	1,559,673	487,401	2,373,879	448,492	2,815,587	19,183	146,565
Mica**	22,014	1,978,157	35,701	963,169	69,650	1,868,544	7,639	3,945,711
Lithium minerals	6,544	241,510	3,334	259,457	12,897	896,000	†	†
Beryl	258	27,150	146	37,361	439	14,500	4,316	1,366,772
Columbium-tantalum ores	3	10,743	2	6,225	‡	1,528	888	1,552,776
Tin	28	28,160	17	32,958			64,822	157,019,209
Total	383,075	3,845,393	526,601	3,673,049	531,478	5,596,159	96,848	164,030,933

* Foreign market value does not include freight to United States, import duty (if any), or other costs of importation.

** Includes scrap mica and mica recovered from kaolin and schists.

† Not reported separately.

‡ Less than 1 ton.

Source of information: U.S. Bureau of Mines.

likely to occur in well-zoned pegmatites, the portions of a pegmatite that contain the most mica are generally less rich in feldspar, and vice versa.

Whereas feldspar is a potential joint product of almost all pegmatite operations, the feldspar industry has not hitherto been a major source of other minerals. A good deal of feldspar is still produced by selective mining, the muck being handled with a fork to reject fines. A substantially growing proportion of the total domestic output, however, is coming from flotation plants processing run-of-the-mine pegmatite or alaskite, and some of these plants produce flake mica and perhaps quartz. Since most of the best feldspar deposits are not rich in other economic minerals, and since other economic minerals, if present at all, generally occur in parts of the deposit which may not pay to work for spar, the feldspar industry is a less important potential source of byproduct minerals than is generally supposed.

Despite occasional shortages caused by swiftly rising demands, mainly in the glass industry, supplies of feldspar in the United States are so enormous that production can respond quickly to any foreseeable increase in demand. In the event that byproduct production of feldspar caused a sharp reduction in prices, demand would probably increase, but further expansion in normal requirements is more likely to pace the general level of industrial production.

Mica: Strategic mica comprises sheet or block mica in the rifted condition (roughly split and trimmed), No. 6 (about 1 by 2 in.) or larger in size, of *stained* or better quality, having suitable electrical properties, and relatively free from cracks, pinholes, reeves, waves, and other mechanical defects. No suitable substitute has yet been found for the fine-grade mica used in high-duty capacitors and vacuum tubes, essential components of automatic measuring instruments, control circuits, and electronic devices used for military and civilian communications. Only the highest quality of mica is acceptable for government stockpiling, and mica from many mines will not meet specifications for use in capacitors set forth in ASTM Designations D-351 and D-748.

Muscovite is of widespread occurrence in common granites and schists, but only in small flakes. Schists composed principally of mica are significant sources of ground mica, and more and more mica is reclaimed as a byproduct of milling kaolin, kyanite, and several other nonmetallic minerals. Strategic mica, however, is found only in pegmatites, and even in the best mines, this is an erratic and minor constituent. The total yield ordinarily ranges from 2 to 7 pct of the rock broken. Moreover, only about 3 to 8 pct of the mine-run mica produced at domestic mines during World War II was finally obtained in the form of full-trimmed block. The yield of strategic mica is frequently less than 1 lb per ton of rock broken.

Following the uncertain and laborious operations involved in finding workable deposits, exploring them for mica-bearing shoots and extracting the mica-bearing rock, there is still a vast amount of work to be done to prepare it for market. Most of the hand-cobbed mica, at least 90 pct and often 95 pct or more, is necessarily rejected in the subsequent rifting, trimming, and splitting operations. It goes to the fabricator in the form of irregular pieces which must be reclassified and re-sorted before going to punching machines. Stampings can

be made only one at a time, the operator holding the small piece of mica in her fingers. Even with careful selection of raw material and efficient operators at the punching machines, at least 70 pct of the weight of the closely trimmed mica sheets is wasted in manufacturing. In short, as little as 0.3 lb of mica stampings is normally obtained from a ton of rock actually mined.

From prospecting to final fabrication and inspection of products, the mica industry thus fails to fit into the modern pattern of mass production. Despite these impediments to the establishment of the industry on a firm foundation, the United States must have mica, and in large amounts. There are now insistent demands for defense production and stockpiling. Prices offered for strategic mica from domestic mines are many times higher than ever before and programs for augmenting supplies from India, Brazil, and certain other important foreign sources are being pushed forward energetically.

Lithium minerals: Although lithium is recovered in significant quantities as a minor constituent of saline deposits, the pegmatite minerals, spodumene, amblygonite, and lepidolite, are most important in meeting the fast-growing requirements of this element and its compounds. In South Dakota and North Carolina there are measured reserves in many tens of thousands of tons and inferred reserves in millions of tons. The numerous large deposits near Kings Mountain, N. C., are remarkably persistent and the spodumene content amazingly uniform, almost invariably exceeding 20 pct. In contrast to conditions at other commercial pegmatites sources, however, the texture of these deposits is such that froth flotation rather than hand picking is necessary to obtain a reasonably clean product. Potential byproducts—in addition to the usual feldspar, mica, and quartz—include small amounts of beryl and traces of the heavy minerals, cassiterite and columbo-tantalite.

Notwithstanding the enormous resources in several states, domestic supplies of lithium minerals usually have been imported. Southwest Africa has been a major source in recent years and now Manitoba has begun to develop its large deposits. Although still rated as a rare metal, lithium is one of the very few useful elements that can be obtained readily from pegmatites in vastly greater quantities, both in this country and abroad.

Beryl: In contrast to the situation with respect to lithium, procurement of beryllium has developed a long series of problems with no prospect of relief. The price lever has jacked up world production more than tenfold since 1937, but this is not enough. Fortunately, the wartime scare abated when graphite was substituted in the atomic stockpile, but prices again climbed rapidly, even before the Korean conflict, with the urgent demand for beryllium alloys for a variety of equipment needed by the armed forces and by civilians. The government now offers to buy clean beryl at \$400 a short ton, following inspection, with the expectation that it may not average more than 8 pct BeO. This compares with a prewar price of \$30 or less per ton for material containing a minimum of 12 to 14 pct. Doubts are expressed as to whether future world requirements can be met if prices go to \$75 or more per short ton unit.

The United States, by far the principal consumer, has always had to seek its major supply of beryl abroad. Domestic production probably never exceeded 100 tons in any year before 1940, and al-

though there was bonanza strike in the Harding mine in Taos County, N. M. in 1950, the all time record annual output in this country has barely exceeded 500 tons as against requirements variously estimated up to ten times this figure.

Much of the world's beryl still comes from the beryl-tantalite mines in the bulge of Brazil. Argentina's output has been small since the famous Las Tapias mine was exhausted in 1941. This is probably the only mine that ever produced beryl as the main product. Stimulated by the short-lived embargo upon exports from Brazil, production was expanded in India and Africa. Currently several African sources seem promising, notably a long belt containing numerous pegmatite dikes in central Nigeria.

In French Morocco the present writer has seen ragged Berbers handcobbing beryl from rock fragments no more than half an inch across and sorting out pieces of beryl scarcely larger than a grain of corn. Even in low-labor cost countries, however, there is a limit to hand picking. There are fairly large known reserves of beryl in the outer zones of numerous domestic pegmatite deposits, but these are very low grade and the crystals so small that they cannot be unlocked except by fine crushing or grinding. Beryl is readily separated by froth flotation from most of its associated minerals except spodumene. Apart from unlocking disseminated grains and obtaining surfaces fresh enough to be filmed by flotation reagents in a pulp which has undergone kaolinization, the main milling problem is the minute volume of concentrates. The mill feed in most cases will not run over 1 pct of recoverable beryllium mineral, so the problem is economic as well as technologic.

The writer has not investigated rumors that beryl occurs in disseminated deposits in unpegmatized or only partially pegmatized rock masses in at least two places in the United States and also in Africa. With these possible exceptions, future supplies of beryllium evidently will depend upon utilization of the exceedingly low grade occurrences of beryl in pegmatite bodies. Some years ago attention was drawn to a New Mexican occurrence of helvite, but further prospecting failed to disclose enough tonnage or grade to permit commercial production.

Columbium-tantalum minerals: There is reason to hope that dependence upon pegmatites for columbium (niobium) and/or tantalum may some day be at least partially relieved. During the late war, the Germans gave attention to deposits of koppite in limestone, one in Norway and a smaller one in Germany. Considerable work was done on the Norwegian deposit after the war, but none of these investigations has resulted in commercial production. Currently, however, attention is being focused on occurrences of minute amounts of columbium in Arkansas bauxite and other deposits associated with nepheline syenites. This element is far more widely dispersed than previously supposed and may some day be recovered from residues of the Bayer alumina process and perhaps from other byproduct sources.

Since tantalite and columbite form an isomorphous series, ores from most localities contain both elements in appreciable amounts. The two elements, however, have different uses; consequently the mixed ores are considered less desirable than those containing a large excess of one or the other

element. Columbite, for example, preferably contains a minimum of 45 pct Cb_2O_5 and the Ta_2O_5 content should be low; some contracts still specify a 10 to 1 ratio. The question of how best to handle mixed ores has not been solved, but the basic problem is to get enough columbium for its rapidly expanding field of use in welding stainless steel and for certain high-temperature alloys. Few pegmatites contain even recognizable traces of these minerals, and microlite, the only other mineral that can be termed an ore of tantalum, is producible only sporadically and in relatively small quantities.

Since domestic production of these minerals has never amounted to much, the United States has depended largely on imports. Hitherto the world supply of columbite has come mostly from Nigeria, where it is a byproduct of tin placer mining, and tantalite has come mainly from Brazil, where it is mined along with beryl. The Belgian Congo has recently become a large supplier of mixed ores which occur in decomposed pegmatites mined for tin. Smaller amounts of mixed concentrates are produced irregularly in other countries but known world sources are inadequate to meet the potentially heavy demand.

Miscellaneous minerals: Quartz, the almost universally abundant constituent of pegmatites, is often saleable. Crystals suitable for radio or other piezoelectric uses are almost never found in pegmatite, but pegmatite quartz is often sought for making crushed grains for coated abrasives and bit-stone or for placing sand for potteries and special refractories. Hand-cobbed quartz has been employed as a substitute for Danish flint grinding pebbles, and coarse tailings from feldspar mills often qualify as glass sand. Crude or roughly crushed quartz is used principally for producing metallurgical flux, for making ferrosilicon, for filling acid towers, and for grinding into silica flour. For most of its uses, however, prices are low, often under \$3 and rarely as high as \$10 a ton.

A variety of gems occur in granitic pegmatites. Among the most important are the beryl group, tourmaline, topaz, spodumene (kunzite and hiddenite), quartz, and feldspar crystals. Rose quartz, smoky quartz, clear quartz, and amazon stone may occur in massive form or in large crystals in the main body of the pegmatite. Although usually outside the main feldspar zone they are recoverable at some feldspar mines as a byproduct. Other gem materials, however, tend to occur only in vugs or fissures or embedded in clay-filled cavities. Owing to their mode of occurrence and more especially to their fragility, gem materials cannot be considered a potential byproduct of ordinary pegmatite mining. Theoretically, many of them might report with other heavy minerals in a mass mining and milling operation, but even if they accompanied the more abundant minerals they might be badly damaged by heavy blasting and by the crushing operations. In mines worked for valuable gems, blasting must be stopped as soon as the gem-bearing zone is reached and further excavation accomplished mainly by chipping, wedging, and prying away enclosing rock. Even heavy sledging may induce the formation of incipient cracks.

Tin, tungsten, zircon, metallic sulphides, several uranium minerals, monazite, and even gold and graphite are minor constituents of various pegmatites, but commercial production of these minerals from pegmatite deposits is insignificant. The minor

elements *caesium* and *rubidium* do come from pegmatites, and in Madagascar and perhaps elsewhere complex yttria-bearing minerals, *gadolinite*, *cyrtolite*, and *fergusonite*, are produced on a small commercial scale as well as for mineralogical specimens.

General Economic Considerations

All who are familiar with problems of mineral supply are aware of the amazing record of the American mining industry in its struggle to hold down costs despite tremendous handicaps. Improved technology and better management have gone far to offset soaring wage rates and the necessity of working leaner and leaner ores at greater and greater depths. As this paper has already emphasized, increased understanding of the structure of pegmatites and the occurrence of these minerals has lessened the risks involved in prospecting and mining. Refinements in froth flotation, magnetic separation, electrostatic processes, and other methods of mineral dressing now permit much sharper separations of the minerals in mixed ores than seemed possible even a very few years ago. Last of all, there is government assistance in financing exploration under the Defense Minerals Exploration Administration. Other agencies advance loans against production and guarantee incentive prices for products.

Nothing is apparently wrong with this economic setting for a relatively small number of favorably situated major pegmatite deposits. For many years the present writer and others have advocated establishment of an enterprise based upon mass mining of pegmatite material, wall-to-wall mining in some cases, followed by suitable milling operations designed to recover all useful minerals from the ore. To test this objective government-financed demonstration plants are now seriously being considered in two or more districts.

Pegmatite operations that can be fitted into the broad pattern of American production should be able to prosper. The real question is how many pegmatite deposits or groups of neighboring deposits can be operated on a large enough scale or can furnish sufficiently uniform flow of mill feed to conform to this pattern. The spectacular technologic advances in mining methods have been made at large mines. Given a suitable mill to treat mixed ores, however, the workings even in small mines can be made wider, and less care need be taken to avoid dilution of ore in laying out stopes. Even in small and spotty deposits where the workings must still be narrow and crooked, the use of sharper drill bits, more efficient rock drills, portable compressors, and better pumps and hoists will help to offset higher wages and other costs. It remains to be seen how great the savings will be.

In the days when efficient labor was very cheap in the low wage areas where so many of the pegmatite mines were situated, it was notoriously true that even the best mines were worked only intermittently by a succession of different operators. The same dike might be worked exclusively for one mineral for a time and then, after a lapse of months or years, it might be worked for another. Fluctuating markets, lack of capital, and above all the seemingly unpredictable nature of these baffling deposits have discouraged attempts at systematic mining. Temporary abandonment may not cause cave-ins that ruin the orebody, but a good many deposits have been ruined by gophering. Absentee owners have been plagued by unauthorized working of their mines, pilfering of tools, and sabo-

tage to buildings. At some operations high-grading is a problem. With the government paying up to \$70 per lb for mica there is a strong temptation to establish a black market for mica bootlegged from legitimate operations. The magnitude of this problem is best illustrated by Indian official statistics which revealed that as much as 2.4 lb of mica were stolen or illegitimately produced for every pound marketed through regular channels.

The most serious threat to the establishment of a successful pegmatite industry that will supply strategic minerals from domestic mines is the mounting difficulty of operating small mines. The latest census data relate to 1939, and the situation has improved very little since then. In that year only four feldspar companies marketed over \$50,000 worth of products. Many feldspar, mica, and other pegmatite mining enterprises were too small to be counted under the census rule, which excluded those producing less than \$2500 worth of products or spending less than \$2500 in their business. The average feldspar miner in North Carolina produced only 0.156 short tons of crude spar for a wage of \$0.28 per hr, and in Colorado the highest state average was only 0.415 tons per man-hour for a wage of \$0.53. For all 59 feldspar mines reporting to the census in that pre-war year, the average value of products per man-hour was only \$0.96; for mica mines it was \$0.91. These figures compared with \$3.95 at iron mines and \$2.76 at copper mines.

By organizing a group of mining units around a central mill and by utilizing old dumps to augment the tonnage from current mining, it may be possible in several districts to develop pegmatite mineral production units of economical size even where no really large deposits can be found. Even with these additions, however, the probable number of larger pegmatite operations is limited and the extent to which they can be depended upon to supply rarer elements as byproducts is debatable. The fact remains that as wages and labor conditions rise to meet higher standards throughout the world, the sources of supply of any mineral that comes principally from isolated small and highly irregular pegmatite deposits become more and more vulnerable. In terms of national policy, this calls for even greater emphasis upon the search for substitutes or ways and means of getting along without certain of these minerals, of which mica and beryl seem to be especially critical.

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St. Joseph Lead's Indian Creek Development



by C. Kremer Bain

DURING the past several years of diamond drilling in Washington County, Mo., the St. Joseph Lead Co. has discovered a concentration of commercial lead-zinc ore at four different points within an area of 36 square miles. The largest and to date the most promising group of these orebodies is located in the Indian Creek area. It was considered to be of sufficient value to warrant development.

The Indian Creek area is in the center of the northern half of Washington County, 11 miles northwest of Potosi, Mo., the county seat. The location is readily accessible from Potosi by car or truck over the all-weather Missouri State Highway No. 114.

The topography is one of rolling relief, varying from an elevation of 700 ft above sea level on Mineral Fork and Indian Creek to a maximum of 1176 ft at High Point Tower. It is divided in equal halves between the drainage to Mineral Fork and to Indian Creek. The shaft site at 1060 elevation is on the divide between the two drainage areas. The Indian Creek side is rough and heavily wooded, with only a few cultivated clearings. The slopes to Mineral Fork are more gentle and rounded, with much of the area cleared for farm land. Most of the surface is covered with Potosi and Eminence formations, which show little evidence of any structure. Ore structure was worked out by extensive diamond core drilling, with holes varying in depth from about 700 to 1300 ft. As of Oct. 1, 1950, 218 prospect holes had been drilled within the 36-square mile area.

The ore is disseminated galena with considerable accompanying sphalerite in the Bonne Terre formation in the shape of irregular blanket-type deposits. The main controlling structural feature of the outlined mineral zone is a buried pre-Cambrian rhyolite-porphyry ridge extending partly through the Bonne Terre formation, which laps against the por-

phyry and also covers it. The mineralized zone parallels and forms an irregular halo around the porphyry ridge and is concentrated in the Bonne Terre formation where the La Motte sandstone is cut out by the porphyry ridge. This is a very common mineral occurrence in the southeastern Missouri lead belt district. The La Motte sandstone, which normally underlies the Bonne Terre formation, is thus cut out entirely. In this area the Bonne Terre is usually about 275 to 320 ft thick, the La Motte varying in thickness from zero up to 400 ft.

Commercial concentrations of ore are indicated at several places, the most notable being at the shaft site where an orebody at least 4000 ft long with widths of 500 to 600 ft has been outlined. The total thickness of mineralized rock reaches 150 ft in some holes, although total ore thickness runs from 8 ft to about 60 ft, often in more than one layer.

Preliminary Planning

Ore reserves as outlined lend themselves to open-stope mining with trackless haulage and mobile loading machines and drilling equipment. This type of equipment has proved to be very efficient during recent years in the company's lead belt operations.

After a study of bids for shaft sinking and an estimation of comparative costs, it was decided that the shaft would be sunk by St. Joseph Lead Co. personnel using company equipment.

Since the plant site is about 26 miles from the company's nearest lead belt operations, it was necessary to plan a complete surface plant, including mill, shops, office, change room, and warehousing facilities.

Immediately following the filing of an application for a certificate of necessity, a headframe was selected for which detailed drawings were already available. The order was placed for fabrication so that it could be delivered and erected in time to serve for sinking the shaft. This eliminated the expense of a temporary headframe.

Drilling was started on two churn drill holes near shaft location, and two 1000-gpm deep well pumps

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Discussion on this paper, TP 3601A, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, Jan. 15, 1953. Los Angeles Meeting, February 1953.

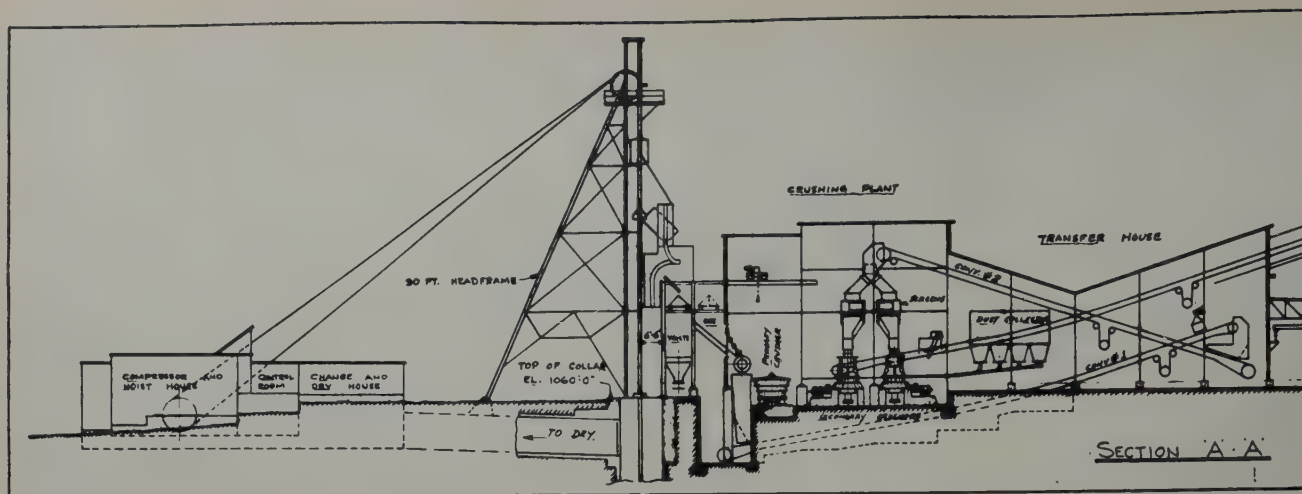


Fig. 1 (above)—The general arrangement of equipment at Indian Creek. Fig. 2 (right)—A plan of the surface layout at Indian Creek.

were ordered for installation in these holes in an attempt to lower the water table ahead of the sinking operations.

Contracts were drawn with Arkansas-Missouri Power Co. for construction of 11 miles of 33,000-v power line from Potosi and for installation of a bank of three 1000-kva, 33,000/440-v transformers at the plant site. Dependable and quick communication being secondary only to power on a project of this size, arrangements were also made for a telephone line from Potosi to the plant site.

Company geologists selected a lead belt stope from which ore was being mined similar to that indicated by the drill cores at Indian Creek. A sizeable sample was taken from this stope for metallurgical and grinding tests to determine the flow-sheet to be used in the new mill of 2000 tons per day capacity.

The nearest railroad to the plant is a spur of the Missouri Pacific which runs from Mineral Point, Mo., to Potosi. Topography of the intervening country makes it impracticable to extend a rail connection to the plant. However, the all-weather graveled highway into Potosi should be adequate for heavy truck transportation. Lead and zinc concentrates will be hauled by truck to Potosi where they will be loaded into cars at the rail head for shipment to the smelter. Operating equipment and supplies will be trucked from Potosi or from the St. Joseph Lead Co. central warehouse located in the lead belt about 30 miles from the plant site.

The topography of the plant site selected permitted location of the shaft very close to the lowest elevation of ore and provided a hillside slope for the concentrator as well as gravity flow for the tailings disposal. Thus fewer pumps will be required. A reservoir located 400 ft from the mill site will furnish water storage at a static head of 80 ft, eliminating the need for mill feed pumps.

Standardization of equipment to conform with newest types of company installations in the lead belt was necessitated by the isolated location of the plant and need for interchangeable spare parts.

Requirements for separation of lead and zinc concentrates called for a differential flotation plant. To minimize equipment requirements and still maintain the advantage of separating free lead in the coarser feed which, in the lead belt, is recovered by gravity concentrating tables, it was decided to use jigs for scalping between the relatively coarse

grinding rod mill, and the classifiers in closed circuit with the ball mills.

A contract was awarded to General Engineering Co., Ltd. of Toronto to design the surface plant and supervise construction. Construction work was subcontracted under supervision of a project engineer and construction superintendent from General Engineering Co. who worked on site. Sub-contracts were given with the provision that the contractor would use local labor whenever possible, so that a favorable company-community relationship would be established. Fabrication and erection of steelwork, as well as plumbing, heating, and electrical work, were given to established St. Louis firms. All tile work was awarded to the local contractor nearest the Potosi area.

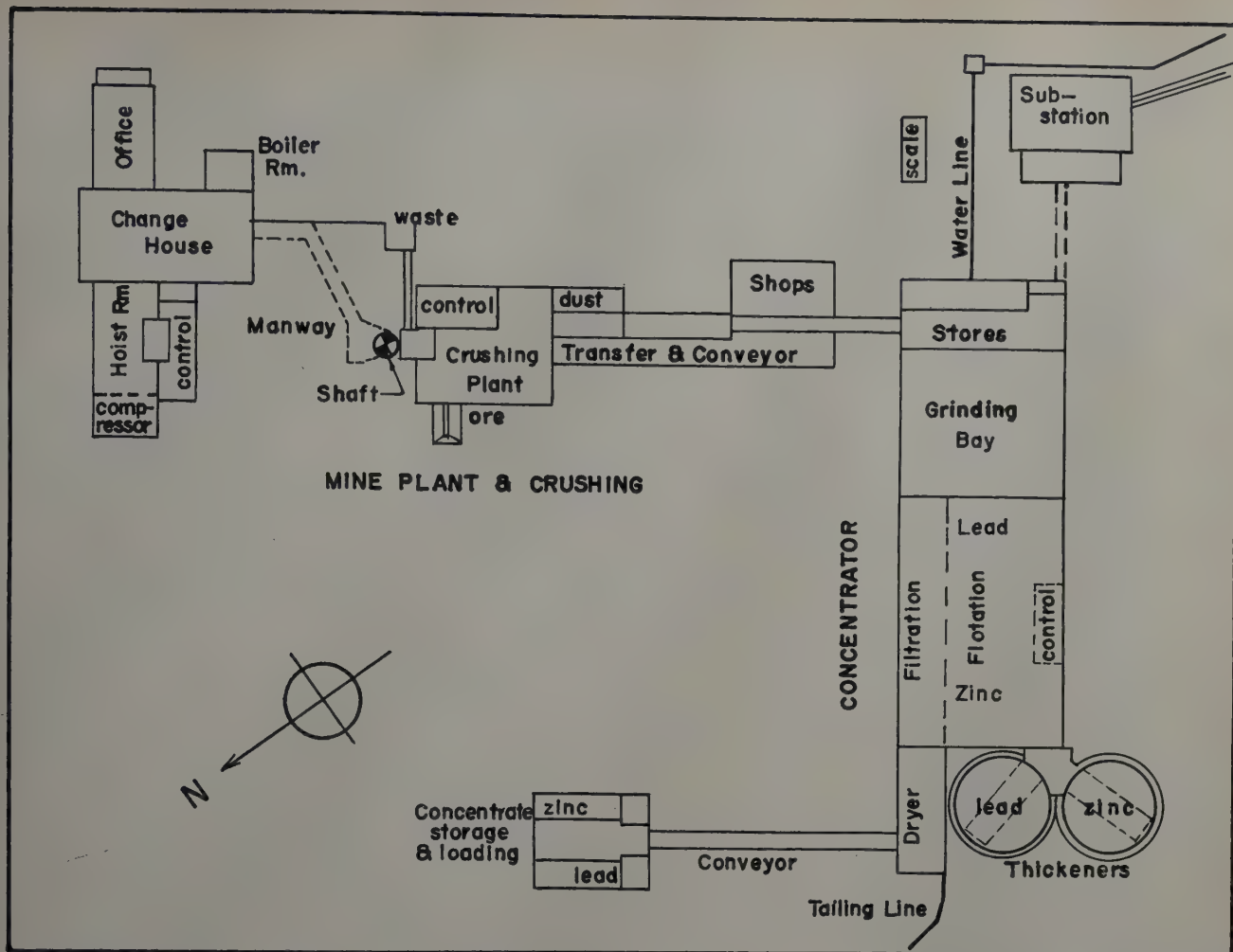
Plant Layout

The frontispiece is an artist's conception of the plant. Figs. 1 and 2 are the surface layout, with location of the two-story office building, change house, heating plant, hoist and compressor room in relation to the shaft, crushing plant, concentrator, concentrate loading stations, sub-station, and reservoir. It is to be noted that the supply warehouse is located immediately under the ore bin, and surface shops under the conveyor gallery in what ordinarily would be wasted space. Thus shop service and supplies are in the center of operations. All buildings are of fireproof construction, with steel framing and Spectra-Glazed tile walls. The tile is a haydite block with ceramic paint baked on one or more surfaces. It can be ordered with different colors on different faces of the tile, so that outside and inside finishes of the building are automatically furnished.

Many modern features were incorporated in the new change room. Outstanding among these are the radiant heating system in the floors and the tunnel connection to a point 20 ft below the shaft collar, making it possible for men to go in and out of the mine without being exposed to the weather. Lockers for street clothes are recessed in the walls, and there are hanging baskets for work clothes. These baskets may be raised to the ceiling by cords attached to individual lockers. The floor is finished with anti-bacterial cement for prevention of athlete's foot.

Hoist

The hoist will have 1½-in. diam ropes overwound and underwound on a 10-ft diam single drum with 8-ton skips hoisting in counter-balance at a rope



speed of 816 fpm. It will be powered by a 600-hp, 2300-v motor and will have a capacity of 350 tons per hr when hoisting from the 920-ft depth.

The loading arrangement at the bottom of the 1500-ton capacity skip pocket is believed to be the first of its type to be used in non-ferrous metal mining operations. Fig. 3 shows a cross section of the general arrangement of the skip pocket and loading equipment. The pan feeder will draw rock from the pocket and discharge into a hopper where the rock will be weighed by an electronic device to the tonnage decided upon for each skip. The operating console will be arranged so that skip loading can be manual or automatic.

Power

Power will be supplied by the Arkansas-Missouri Power Co., which in turn receives power from a Union Electric Co. sub-station at Shirley, Mo. This new 10-mile power line from the sub-station connects with the St. Joseph Lead Co. transformer station at plant site. The transformer station will be an outdoor packaged unit complete with structure, bus, wire, hardware, primary air switches, fuses, lightning arresters, 7500-kva, 33,000/2300-v transformer, and secondary breakers. It will furnish 2300-v power to the shaft and to distribution centers in the hoist room, crushing plant, grinding section, and flotation section. All motors over 150-hp will be 2300-v and smaller motors will be 440-v. The 2300/440-v transformer will be located at or near load centers. Distribution will be from central metal-clad power control centers with pull-out type, removable circuit breakers.

All control centers will be located in an air-conditioned dust-free room, and all motors will be controlled by push buttons located near the operator's position, with interlocking controls where necessary to insure proper sequence of starting and stopping. E. H. Tucker of Newmont Mining Corp. designed the electrical layout.

Mill

The graphic flowsheet, Fig. 4, shows size and type of crushing, grinding, and flotation equipment to be used for handling 2000 tons per day. The crushing section has a capacity of 2000 tons in 6½ hr, the same as that of the hoisting equipment. Storage bin facilities of 2200 tons will insure ample feed to the grinding and flotation section for the 2000 tons per 24 hr. If future conditions warrant expansion of this mill, to double plant capacity it will only be necessary to duplicate the grinding and flotation section and go to two shifts hoisting and crushing.

Shaft Sinking

It was decided that a circular shaft with 12-in. thick concrete wall would give the strongest section through the upper Potosi formation, which showed every indication of being heavily water-bearing. The shaft also would provide maximum clearance with the least amount of excavation to accommodate hoisting and lowering of large trackless haulage equipment. An ID of 12 ft 7 in. would allow for handling of 8-ton skips plus thickness of guides and supports.

Years of experience in operations have proved that when machines are simply and properly designed even inexperienced crews, under supervision,

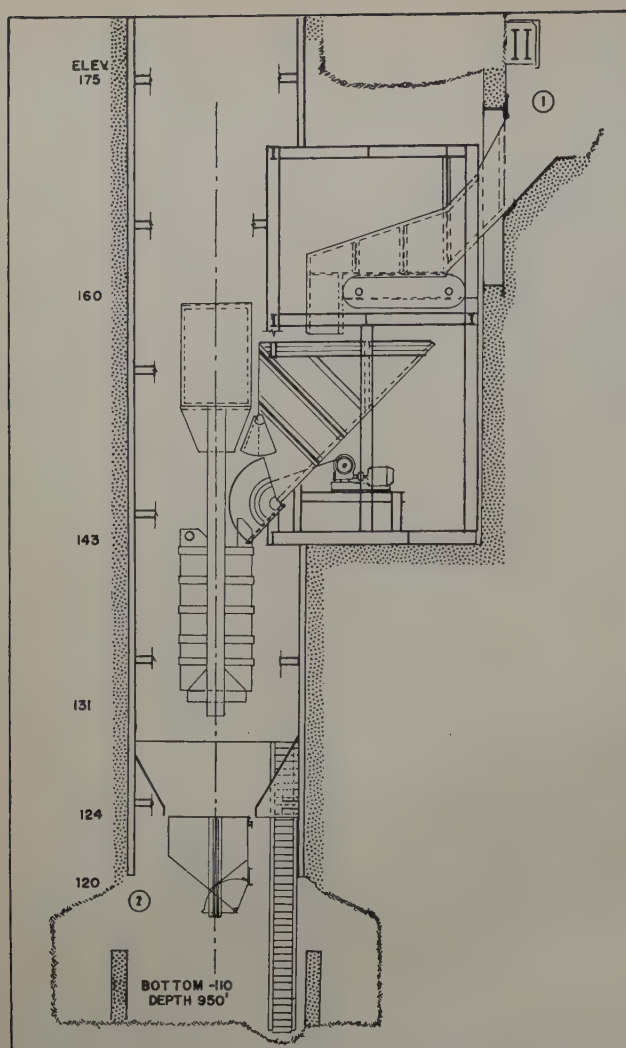


Fig. 3—Arrangement of the skip pocket. Circled numeral 1 indicates 1500-ton capacity skip pocket. Numeral 2 indicates point at which shaft concreting stops, 940 ft from the top of the shaft.

can sink a shaft safely and efficiently. A number of pieces of equipment were designed specifically for sinking the Indian Creek shaft. When possible, plans were obtained for still other pieces of equipment found suitable on similar shaft-sinking operations in other districts.

Final plans called for sinking of a cylindrical shaft 950 ft in depth and about 15 ft in diam. With the exception of landing stages and the bottom 10 ft, the entire shaft was lined with concrete to an ID of 12 ft 7 in. One 10-in. pipe and four 4-in. pipes were placed in the concrete lining in one quadrant. Thus located in the concrete lining, these pipes would not only be out of the way during future hoisting operations but also could be used in the actual sinking operations for compressed air, ventilation, pump discharge, and many other purposes.

A circular, four-sectioned steel form was designed for use in pouring this concrete lining in increments from 5 to 20 ft. This steel form remained in place on the last segment of lining poured until sufficient shaft was sunk for another section. A winch and drum containing 1000 ft of $\frac{5}{8}$ -in. steel cable was installed on a concrete pier at surface and a sheave installed on the inner lip of the shaft collar for each of the four sections of the steel concrete forms. By this arrangement, the forms were easily lowered

into position for pouring the next segment of the shaft lining.

Convenient, yet out of the way, these steel concrete forms protected the green concrete lining from both fragments and blast effect. Sufficient drill holes and powder were used to insure relatively fine fragmentation to reduce further the possibilities of damage to the shaft lining and other installations. Concrete lining was carried as close as possible behind the sinking crew to protect the men from any possible rock falls or cave-ins and to minimize scaling. In addition, ground water was sealed off as sinking operations advanced.

A column of 6-in. H-beams, located at each of the four quadrant points, was designed to serve as bracing for the steel concrete forms and to act as guides for the work cage and other pieces of equipment. After completion of the shaft-sinking operations, these H-beams will serve as supports for actual skip guides. They were prefabricated, drilled to a template, and anchored by $\frac{3}{4}$ -in. hook bolts set in the concrete lining. A template, wing forms, and fan tail jacks were used to locate the H-beams in position where they could be jacked solid after being plumbed.

A double-decked steel work cage was designed for shaft-sinking operations, with guide shoes engaging each of the four H-beam columns. This cage not only served as a platform from which the crew could work at any point in the shaft, but also provided facilities for lowering equipment and materials, and acted as internal bracing for the forms as the concrete was being placed.

All electrical control wires were suspended from a steel cable and connected to a canopy. These included wires for lighting systems, speaker system, signal system, electric hoist controls on the work cage, and mucking machine controls. Outlet plugs for various control wires, a speaker, and lights were installed on this canopy.

A positive displacement Roots-Connorsville blower, designed to supply air at 4000 cu ft per min at $2\frac{1}{2}$ psi, was connected to the 10-in. pipe set in the concrete lining. This blower furnishes ventilation for the shaft and subsequent operations until permanent ventilation is established.

Also used in sinking operations were customary items of equipment such as grout and sump pumps, compressors, drill presses, electric and acetylene welding sets, and a concrete batching plant to insure a proper and uniform mix. Concrete was delivered from the mixer to a $1\frac{1}{2}$ cu yd bottom dump concrete bucket by means of a 24-in. trough conveyor. Washed sand and gravel were stockpiled near the batcher. Water was supplied by a deep well pump, and cement was stored under shelter convenient to the batcher.

A suitable drill round with all holes properly located was of great importance. Even the most experienced drill crews often failed to spot and drill holes of a round accurately. This is considered to be a major factor contributing to misbreaks. A four-machine jumbo drill was therefore designed for the Indian Creek shaft. Each machine was independent of the others. A fixed angle between the feed and the arms, in conjunction with a special indexing device, provided a rapid and accurate

Fig. 4—The graphic flowsheet on the righthand page lists the size and type of equipment in use at the St. Joseph Lead Co. Indian Creek development.

method of aligning drill steel and spotting the hole with proper pitch and depth.

A 56-hole round of drill holes, each 1½-in. in diam, was used for breaking the shaft. By means of the special features on the jumbo, the drills could be set for breaking a theoretical 8, 9, or 10-ft round. This particular shaft round might be classed as a modified conical cut with two rings of relief and one ring of line holes. By means of the jumbo, all holes could be drilled accurately with cuts actually meeting at a point, and line holes could be drilled to minimize overbreak.

Mucking the round in a shaft has generally been considered one of the toughest and most time-consuming jobs in sinking operations. Clamshells had been used successfully on other mucking jobs, so a mucking machine with the clamshell powered by two air motors with spring-loaded brakes was designed for this operation. Motors were located on the surface and operated by remote control from the bottom of the shaft. By this means, the operator had complete control of the clamshell from a position where he could closely observe its operation. The muck was hoisted in a 54-cu ft bucket and dumped by means of a hinged dumping door, while the shaft men were protected by a sliding shaft door. A scraper was used to drag waste rock from the vicinity of the shaft and dispose it in a nearby ravine. Plans for these doors and bucket were ob-

tained through courtesy of the Tennessee Copper Co., Ducktown, Tenn.

The shaft-sinking crew consisted of a general foreman, one shaft foreman, one hoistman, one maintenance man, and four shaft men on each of three shifts. Three shaft men on each crew were recruited from the vicinity and had had no previous experience in mining. None was afraid of work and all were willing to learn. The operation and functioning of the equipment were new to all members of the crew, since it was the first time that much of it had ever been assembled on any job.

Weekly inspections were made of ropes, sheaves, hoist, and compressors, by a machinist from the shop organization in the lead belt. The regular shaft crew handled all maintenance work and breakdowns.

Summary

It is worthwhile to point out that results of company planning were satisfactory both as to cost and progress. Sinking was started with an inexperienced crew, a number of whom had never been underground. The shaft was bottomed by a smooth-working team of experienced shaft men. Their outstanding record of completing the entire 950-ft depth, averaging 7 ft 4 in. per blast without a single mis-break, with excellent fragmentation, and without a single lost time accident speaks for itself.

Technical Note

Calcined Cold-Precipitated Hydrated Iron Oxide

by William A. Mitchell

AN X-ray diffraction pattern for "calcined cold precipitated ferric oxide" is reproduced diagrammatically along with data for other iron oxides by R. C. Mackenzie.¹ This pattern, which shows spacings higher than those for any other form of iron oxide, was obtained from material prepared and photographed at several different times over a period of weeks, but later attempts to reproduce the results were unsuccessful, the pattern of hematite being obtained every time.²

In an attempt to solve this problem, the original photographs showing the strange diffraction pattern were examined and a marked similarity to the hematite pattern was noticed, suggesting that the pattern was in fact due to hematite but corresponded to a shorter wavelength of X-rays than the Cu K α which was assumed to have been used.

The eight strongest lines were provisionally identified with the strongest hematite lines and, by means of the corresponding hematite spacings, the X-ray wavelength was calculated for each line. This gave an almost constant value, with the exception of the innermost line which was somewhat diffuse, and confirmed the presence of hematite. The value obtained, $0.703 \pm 0.015\text{\AA}$, corresponds to the molybdenum K α wavelength.

The X-ray tube used was a four-filament demountable tube with a copper target. Other photographs, mostly of soil clays, taken during the same period gave normal patterns of Cu K α radiation with one exception. This was a calcined soil clay with a high iron content and also showed Mo K α lines of the hematite structure.

The explanation is almost certainly that the copper target was contaminated with molybdenum, but it is not known how this occurred. Specimens of aluminosilicates gave their normal Cu K α pattern, but as the K absorption edge of iron is a little longer than the Cu K α wavelength, specimens with a high iron content greatly reduced the intensity of the Cu K α diffraction lines relative to that of the lines due to the shorter Mo K α wavelength.

The formation of hematite on calcining cold precipitated hydrated ferric oxide³ is thus confirmed.

References

¹ R. C. Mackenzie: Investigations on Cold-precipitated Hydrated Ferric Oxide and its Origin in Clays. Problems of Clay and Laterite Genesis, p. 69, Fig. 4, F. AIME Symposium, St. Louis, Mo. Feb. 19-22, 1951. Published 1952.

² R. C. Mackenzie: Ref. 1. Note added on proof.

³ W. C. Hansen and L. T. Brownmiller: Equilibrium Studies on Alumina and Ferric Oxide and on Combinations of These with Magnesia and Calcium Oxide. *American Journal of Science* (1928) 5, No. 15, pp. 225-242.

W. A. MITCHELL is a member of the Macaulay Institute for Soil Research, Aberdeen, Scotland. TN 176H. Manuscript, May 20, 1953.

Two-Way Belt Conveyor Transportation

by C. W. Thompson



Fig. 1—Simultaneous haulage of coal and supplies.

The two-way belt conveyor for coal mine service simultaneously carries coal from faces and transports men and supplies into the mine, largely eliminating the necessity for rubber-tired supply and man haulage equipment and the necessity of maintaining additional headings.

SINCE 1920 use of belt conveyor transportation has increased steadily, and capacity in tons per hour per unit has increased at about the same ratio. Belt conveyor systems for simultaneous two-way material transportation have been widely discussed, but heretofore have proved impractical for specific operations.

Studies of a proposed means of disposing of washery waste on the return trip of a main haulage conveyor, as yet uninstalled, led to the application of two-way belt haulage to Weirton's underground supply and mantrip problems.

Except for a single-tracked main haulage entry, Weirton mine is essentially a full-belt haulage operation with attendant problems of supply and mantrip haulage that could not be handled with conventional belt conveyors during the regular work shift. It was necessary, therefore, to maintain rubber-tired haulage equipment and haulageways serving multiple-shift continuous operations.

The conventional underground belt conveyor is primarily intended to transport coal from working faces to surface. Belts are usually reversed to haul men to face areas at the start of the working shift or to transport supplies.

The two-way belt conveyor for coal mine service is not yet offered for sale as a unit by any manufacturer and must be built on order. This conveyor, with return run traveling parallel but opposite in direction to the outgoing or coal haulage run, permits not only haulage of coal from faces, but simultaneously permits transportation of men and supplies into the mine at any time, see Figs. 1 and 2. This eliminates, to a great extent, the necessity for

rubber-tired supply and man haulage equipment and the maintenance of additional headings. Unlike conventional installations the return belt is in the open for inspection and maintenance. It is less subject to edge wear, since it runs on regular troughing idlers, see Fig. 3. The only points where the edge of the belt can rub are at the drive unit and the tail units where improved design should eventually eliminate edge wear entirely.

Individual conveyor sections are lighter, since all that is needed is a tubular stringer on each side to carry troughing idlers only, no cover plates being necessary. The return run is carried on an assembly identical to that carrying the coal haulage run. These assemblies are definitely easier to handle and are cheaper than installations for the same length of conventional belt conveyor.

The four pipes required for the complete conveyor frame may serve only as supports, or, with proper couplings, may be used for mine water discharge line, pressure line for face spraying, telephone cable conduit, or any other purpose requiring a pipe line, see Fig. 4.

Additional expense, if any, for the increased number of pulleys needed to turn the belt over for the return service is more than offset by reduced expense for rubber-tired supply and mantrip haulage.

The two-way belt haulage system is confined to a single heading on intake air, isolated from all other headings by stoppings on each side, and open for air circulation only at the main haulage way and at the faces. One additional heading, also on intake air, is maintained as an emergency rubber-tired haulage way, all other headings being center-posted and used as airways only until retreat work begins. This method, of course, can and should be used with any belt conveyor, since it provides two independent intake airways for escape in case of fire on the belt or elsewhere. Belt heading is heavily timbered even under good top conditions and space between the two runs permits center posting for the full length

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Discussion on this paper, TP 3560F, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, Feb. 5, 1953. Los Angeles Meeting, February 1953.



Fig. 2—Mantrip and simultaneous haulage of coal.

of the conveyor system. Top or bottom is taken when necessary at the points where vertical pulleys required for the quarter-turn twists are to be installed. The units where the twist is made are collapsible, Fig. 5, so that they may be moved under low top when the belt is extended. The twist units are installed in the same places on retreat as on advance so that further top or bottom brushing is unnecessary.

Each new installation incorporates improvements brought about by previous experience. The following data from a manufacturer's development problem report presents methods of determining the distance needed for a quarter-turn twist, Fig. 6, a major consideration in the design of the two-way belt conveyor.

The distance required to make the quarter-turn twist will vary as the duck weight, unit tension, and belt width. This report covers 28-ounce, 32-ounce, 42-ounce, and 48-ounce duck with tensions and widths which seem to be practical for a two-way conveyor.

The chief problem in determining distance for the quarter-turn twist was to find a modulus of elasticity for the various ducks. Tests previously made on these ducks were reviewed and stress-strain charts were plotted. Two curves were drawn for each duck weight, these curves being the extremes of the scatter band. The high curve on each chart is a high modulus of elasticity which, in the case of the

two-way conveyor, is the most undesirable and the modulus of interest. For the purpose of discovering whether or not a belt used in a quarter-turn twist would distort as calculated from the modulus of elasticity, an 18-in., 4-ply, 28-ounce belt was set up in the laboratory, Fig. 7. Increments of 23 in. were measured off with no tension; then a known tension was put on the belt and the modulus of elasticity was calculated. This modulus was found to fall below the scatter band of the stress-strain curves made from older tests.

Another problem was to determine whether or not the belt between the horizontal pulley and vertical pulley absorbed the stretch required to make the quarter turn, or whether the belt relayed some of the elongation past the pulleys until some unknown length absorbed it. From the quarter-turn drive set up in the laboratory, it was found by measurement that this elongation must be absorbed between the horizontal and vertical pulleys. This test also proved that minimum tension was at the belt center, increasing on a straight line to a maximum tension at the edge. By means of modulus of elasticity, belt widths, and various belt tensions, calculations were worked out to determine distance required for the necessary quarter-turn twist for a two-way conveyor.

The following method is used to calculate distance required for the quarter turn, see Figs. 8 and 9.

T_s = Tension at edge of belt in the quarter turn drive, lb per ply in.
Edge of belt travels helical path in making quarter turn.

T_m = Tension at center of belt, lb per ply in.

T = Total tension on belt, lb.

L = Center to center distance of vertical and horizontal pulleys, in.

Σ = Change in unit elongation from edge of belt to center of belt, in.

ϵ = Total elongation of edge with respect to center, in.

ΔT = $T_s - T_m$



Fig. 3—A view of the experimental underground two-way conveyor.

E = Modulus of elasticity

W = Width of belt, in.

$$WTm + \frac{W}{2} (Ts - Tm) = \frac{T}{\text{No. of Plies}} \quad [1]$$

Fig. 9 is a linear presentation of Eq. 1.

T is known from requirements of the belt. Tm has a minimum of 0, and Ts a maximum depending on the weight of duck used. The maximum allowable tension for various duck weights is as follows: 28 ounce, 34 lb per ply in.; 32 ounce, 39 lb per ply in.; 42 ounce, 49 lb per ply in.; 48 ounce, 59 lb per ply in.

$$\Sigma = \frac{\Delta T}{E} \quad [2]$$

Since the path of any point on the edge of the belt in the quarter turn is a helix, the solution can be found by the triangular method whereby the helix is the hypotenuse of the triangle, the distance between pulleys the longer side, and the pitch and shorter side one half the belt width.

$$L + \epsilon = \sqrt{L^2 + \left(\frac{\pi W}{4}\right)^2} \quad (\text{For } 90^\circ \text{ turn})$$

$$\epsilon = \Sigma L$$

By substitutions and simplifying:

$$L = \frac{\pi W}{4} \sqrt{\frac{1}{\Sigma(2 + \Sigma)}} = 0.785W \sqrt{\frac{1}{\Sigma(2 + \Sigma)}} \quad [3]$$

The above solution can be used only when the quarter turn is symmetrical as in Fig. 8. Fig. 9 shows a quarter-turn drive that is not symmetrical, and with this drive the following equation must be used. See also Figs. 10 and 11.

T_{SL} = Tension in the low tension edge of the belt in the quarter turn drive, lb per ply in.

T_{SH} = Tension in the high tension edge of the belt, lb per ply in.

T_{MIN} = Tension in the minimum tension section of the belt, lb per ply in.

L = Center to center distance of vertical and horizontal pulleys, in.

y = Distance from high tension edge to minimum tension section, in.

x = Distance from low tension edge to minimum tension section, in.

ΣH = Change in unit elongation from high tension edge to minimum tension section.

ΣL = Change in unit elongation from low tension edge to minimum tension section.

$$\Delta T_H = T_{SH} - T_{MIN}$$

$$\Delta T_L = T_{SL} - T_{MIN}$$

E = Modulus of elasticity.



Fig. 4—Two of the four pipes supporting the conveyor frame are shown here. One pipe carries water in the mine for spraying, another pipe carries mine water out of the mine, the third contains air for roof bolting, and the fourth carries telephone line.



Fig. 5—Here there is a turnover in 46-in. coal without the necessity of taking any roof.

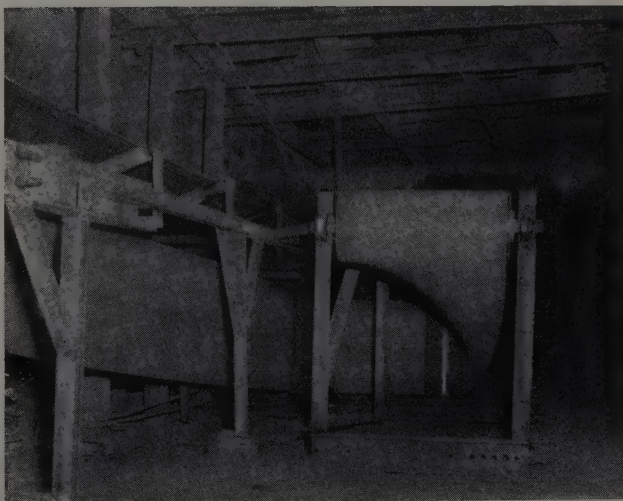


Fig. 6—The quarter-turn twist on the experimental underground unit.

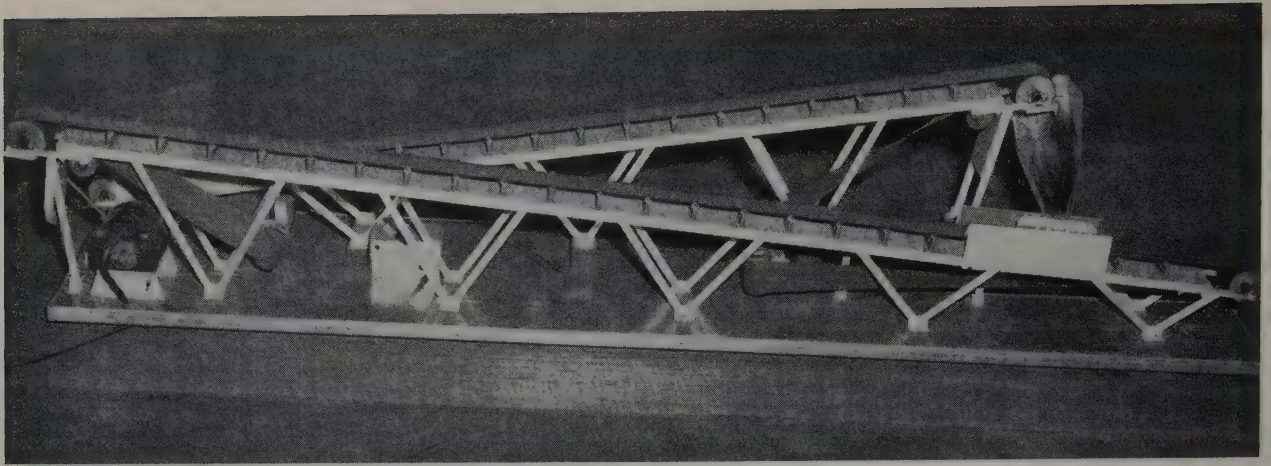


Fig. 7—The experimental unit in the mine administration building.

W = Width of belt, in.

T = Total tension of the belt, lb.

$$WT_{MIN} + x \frac{(T_{SL} - T_{MIN})}{2} + y \frac{(T_{SH} - T_{MIN})}{2} = \frac{T}{\text{No. of plies}} \quad [1-A]$$

$$\frac{y}{T_{SH} - T_{MIN}} = \frac{x}{T_{SL} - T_{MIN}}$$

$$T_{SL} = \frac{x(T_{SH} - T_{MIN}) + yT_{MIN}}{y} \quad [1-B]$$

Substitute this value for T_{SL} in Eq. 1-A. The result is Eq. 1-C.

$$WT_{MIN} + \frac{(x^2 + y^2)(T_{SH} - T_{MIN})}{2y} = \frac{T}{\text{No. of plies}} \quad [1-C]$$

By using Eq. 1-C it is possible to determine T_{SH} and T_{MIN} . First substitute allowable duck strength in pounds per ply inch for T_{SH} and solve for T_{MIN} . If T_{MIN} is 0 or greater, then the values of T_{SH} and T_{MIN} are satisfactory. However, if T_{MIN} is less than 0, it is then necessary to set $T_{MIN} = 0$, and solve for T_{SH} . In determining L , T_{SH} and T_{MIN} are critical factors. This may be stated because T_{SL} will never be greater

than T_{SH} nor will it be less than T_{MIN} . In each case, the modulus of elasticity used is the high modulus as determined from the stress-strain charts. It is felt that the modulus of elasticity used in making the calculations for Table I will be sufficiently high for conveyor belts now in factory production.

$$\Sigma = \frac{\Delta T_H}{E}$$

$$L + \epsilon = \sqrt{L^2 + \left(\frac{\pi y}{4}\right)^2} \quad [2-A]$$

$$\epsilon = \Sigma L$$

By substitution and simplifying:

$$L = \frac{\pi}{y} \sqrt{\frac{y^2}{2\Sigma + \Sigma^2}} = 0.785y \sqrt{\frac{1}{\Sigma(2 + \Sigma)}} \quad [3-A]$$

Table I. Distances Required For Twisting Belts

	Width, In.		Tension, Lb Per Ply In.*				
	18	24	30	36	42	48	60
28 Ounce Duck	10	126**	15	20	25	30	35
	18	96	102	106	133	199	265
	24	167	136	141	177	221	331
	30	209	171	177	221	265	398
	36	251	204	212	248	309	464
	42	292	239	248	309	442	663
32 Ounce Duck	15	96	20	25	30	35	40
	18	129	114	133	166	208	274
	24	161	143	166	208	249	374
	30	193	171	200	249	291	437
	36	225	200	234	291	332	500
	42	257	228	266	332	415	624
42 Ounce Duck	20	92	25	30	35	40	45
	18	123	112	126	147	183	239
	24	153	140	157	184	220	275
	30	184	168	189	220	257	311
	36	214	196	220	257	294	367
	42	245	224	252	294	337	458
48 Ounce Duck	25	87	30	35	40	45	50
	18	116	108	119	133	155	194
	24	145	135	148	167	194	233
	30	174	162	178	200	233	272
	36	203	189	208	234	266	310
	42	232	216	237	266	303	388

* Unit tension in straight portion of the belt approaching the drive.

** An upturn in length of twist at low tension is caused by the need of avoiding compression in the center of the belt, rather than by high edge tension.

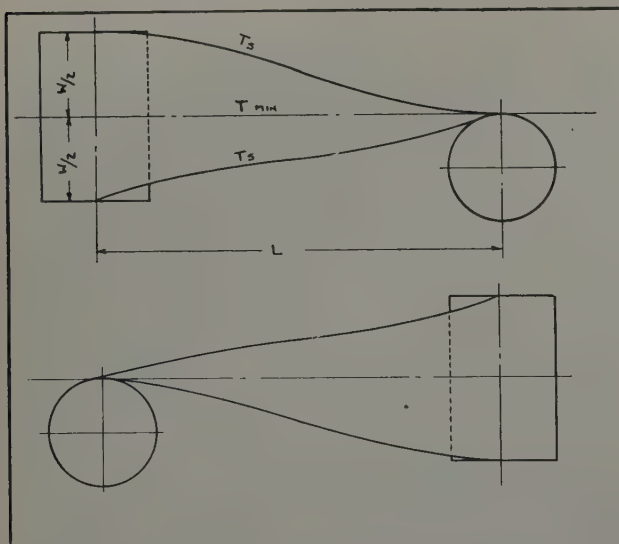


Fig. 8—Sketch of a quarter-turn set-up, symmetrical. Side view shown here, top view below.

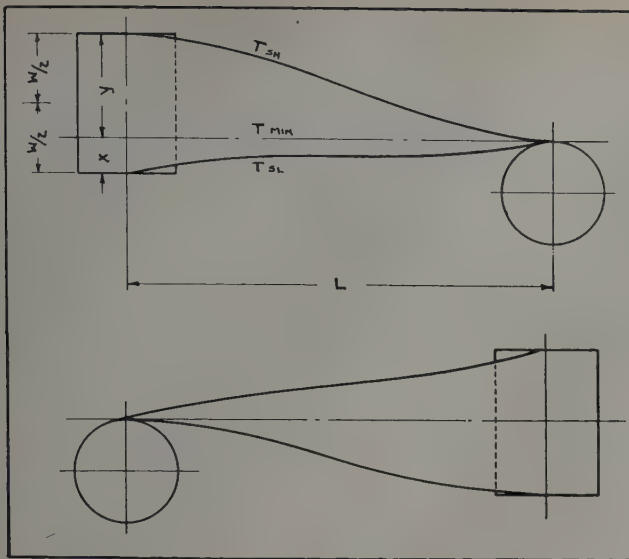


Fig. 9—Sketch of quarter-turn set-up, not symmetrical. Side view shown above, top view below.

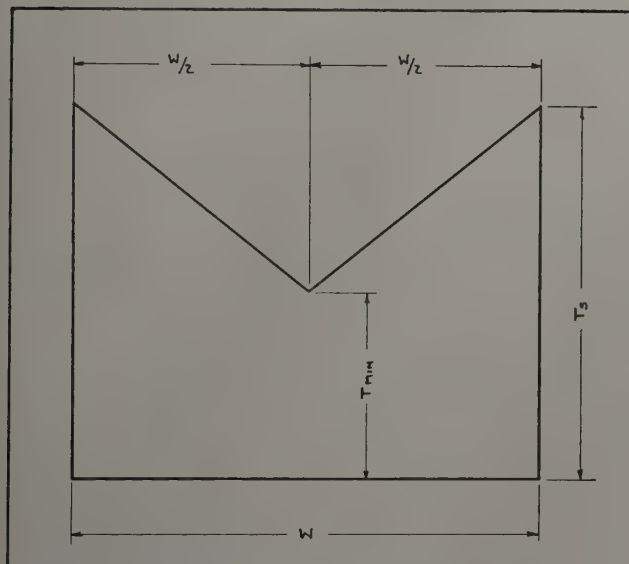


Fig. 10—Tension distribution across width of belt for a quarter-turn drive, symmetrical.

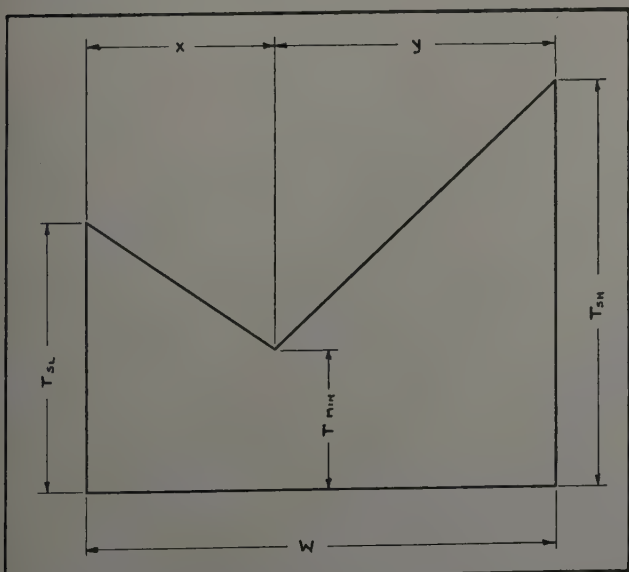


Fig. 11—Tension distribution across width of belt, not symmetrical.

Table I shows the distance required to make the quarter turn for various duck weights, belt widths, and belt tensions.

It is apparent that the twist required of the belt in the two-way conveyor can be made in a distance that is entirely reasonable. The only other problem, from the belt standpoint, is that of training the belt. This is shown to be practical by the laboratory set-up and by existing quarter-turn drives. From the standpoint of equipment there will be, in some cases, a problem of sufficient head room to stand pulleys on end for the twist in the belt and a problem providing additional structure to support the return belt. Experience at Weirton mine has proved that the problems are not difficult to solve.

Belt manufacturers' calculations show the two-way belt to be practical for lengths up to one mile with ordinary belting. For greater lengths a heavier belt is advisable. As an alternative, the conveyor system may be broken up into several flights of any convenient length.

It is not claimed that the two-way conveyor is the answer to all haulage problems in complete belt mining. There will be many cases where it will not even be practical, as in a mining section of very limited length or in an area where tracked supply and mantrip haulage is already installed. However, for mines with large coal reserves, whether in low seam or not, it may be found that two-way belt haulage is applicable, depending on mine layout and extent of development. In many cases where the two-way conveyor can be used, one indicated result will be a reduced unit haulage cost in both directions in addition to a continuous flow of material in both directions without the surges in power and delivery which accompany any system of tracked or rubber-tired haulage.

Summary

As a result of the Weirton study and mine installation, it was found this system accomplishes the following:

1—Rubber-tired equipment for man trips and supplies, cost and maintenance of such equipment, and the necessity for operating personnel are eliminated.

2—Tubular supports for idlers can be used for such purposes as air lines, wires, cable, water, or other carrier lines necessary in modern mines. In many cases this will eliminate the purchase, and installation and maintenance costs, of rib or roof support equipment for these carrier lines. This also applies to pipe for air and water.

3—Time is saved in personnel travel and material handling.

4—There are no empty returns to surface for more men or supplies. Costly driving, grading, supporting, and maintenance of multiple entries formerly needed for these duties are eliminated.

5—Replacement costs are lowered because initial cost of equipment is less.

6—Belt life is increased by reduction of edge wear.

The two-way belt conveyor, therefore, shows promise of improving underground belt transportation because of its multiple uses. However, continued operation under actual mine conditions is necessary to establish the economics on a comparable basis with costs for present-day conventional belt conveyors.

Some Fundamental Principles Applied to the Design And Operation of a Fine Anthracite Plant At Coaldale Colliery

by W. T. Turrall and M. J. Cook

A discussion of modern developments in beneficiation of fine sizes of anthracite, this paper includes a description of the plant flowsheet, an analysis of operating results, and a summary of fundamentals of mineral separation relative to plant operation.

DURING the past several years Lehigh Navigation Coal Co. has installed equipment at Tamaqua and Lansford collieries for cleaning and sizing No. 4 buckwheat $-3/32 + 3/64$ in. and for recovering and cleaning flotation coal $-20 + 200$ mesh at Tamaqua colliery. Papers describing some of these installations have been presented to AIME. At Coaldale colliery, however, methods used to clean fine coal were inadequate and inefficient. Management decided upon a thorough investigation of all existing processes before building a plant to treat the combined three sizes mentioned.

Investigation, carried out on company property and at other plants, entailed a study of hydrotators, hydroclassifiers, tables, spirals, flotation machines, methods of screening, and methods of dewatering. On the basis of this investigation a flowsheet was designed which necessarily excluded many processes in use, and the writers wish to emphasize that no criticism is implied by omission of processes or machines. A determining factor in selection of many machines was the policy of standardizing equipment wherever possible.

Flowsheet and Operational Data

The first principle to be considered in designing of any plant is efficiency of cleaning as related to laws of classification. Beneficiation of coal may be described as separation of two materials having different specific gravities. Although there is an inherent ash content even in the purest coal which slightly affects its specific gravity, it is recognized that many particles are true middlings. The percentage of these particles allowable in the final product

depends upon the ultimate ash, or specifications of the consumer.

Richards in his study of free settling and hindered settling classification¹ determined that particles of equal size and different specific gravities had unequal settling velocities. He also determined a definite size relationship for equal settling velocities with free settling and hindered settling conditions. Considering the design of machines used to clean No. 4 and No. 5 buckwheat, it may be assumed that both free settling and hindered settling conditions exist and that ratio of size for practical operations is between the established values. The essential factor is that the more efficiently and closely sized the feed is maintained, the more nearly perfect the resulting separation.

Flowsheet design was governed primarily by factors related to the 1200 to 1500 tons per hr feed to the main plant. It should be stressed that because of the many sources of supply from mine and stripping, the physical characteristics of this feed varied considerably. The most significant influence was varying percentage of fines, necessitating that all machines be capable of operating under maximum peak load during periods of surge. Since cleaning practice is a wet process, about 9000 gal per min containing 14 to 15 pct solids, or 350 tons per hr, must be treated. This slurry contained all the fines, or $-3/32$ in., but also percentages of oversize, up to $1\frac{1}{2}$ in., dependent on spills or breaks in screen jackets, a not uncommon occurrence in preparation plant practice.

The flowsheet developed is shown in Fig. 1. Circled numbers represent points of sampling. Table I gives operational data including size, ash by size, composite ash, rate of flow in gallons per minute, percent solids dry weight, and average tons of solids per hour. It should be noted that these results are subject to human error in sampling practice. Detailed information reported is a representative evaluation of plant operation.

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Discussion on this paper, TP 3608F, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, March 26, 1953. Fuels Conference ASME and AIME, Philadelphia, October 1952.

Table I. Analyses of Samples Taken from Coaldale New Fine Coal Plant, June 26, 1952

Sample Point on Flowsheet	1* Feed to 24-Ft Hydroclassifier Tank (Calculated)		2 Overflow 24-Ft Hydroclassifier Tank		3 Underflow of 24-Ft Tank, Feed to Over- size Launder Screen (4 Mesh)		4 Oversize from 4-Mesh Screens Re- turned to Main Breaker		5 Feed to No. 4 Buck Launder Screens (12 Mesh)		6 No. 4 Buck from 12-Mesh Launder Screens, Feed to Wemco Spiral	
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8	1.4	35.76			2.0	35.76	36.8	25.40	0.2		1.5	18.97
10	6.7	28.72	0.4	13.01	9.2	29.01	41.9	27.19	6.5	17.80	22.4	25.21
14	7.5	26.02	1.4		10.0	26.71	10.0	21.95	9.7	19.25	30.2	24.39
20	8.9	24.96	1.4		11.9	25.44	2.8	17.99	11.4	18.50	25.6	22.74
28	11.6	23.97	3.4	15.02	14.8	24.79	1.5	15.67	15.4	20.93	9.4	20.43
35	10.0	23.39	4.0	15.93	12.2	24.39	0.8	13.94	13.4	21.95	2.3	16.06
48	8.1	24.91	4.4	16.53	9.5	26.49	0.7	14.68	9.6	25.24	1.2	14.46
65	6.6	22.11	5.6	15.71	6.9	24.15	0.8	15.20	7.4	24.36	1.1	17.12
100	7.3	21.73	9.6	15.71	6.2	25.31	1.1	18.02	7.0	23.03	1.0	16.15
150	5.2	22.09	8.8	17.63	3.7	25.85	0.2	20.80	4.0	26.23	0.7	19.26
200	4.2	23.76	8.0	20.64	2.6	27.00	0.8	24.61	2.8	26.24	1.1	22.25
-200	22.5	36.12	53.0	36.18	11.0	35.49	2.6	35.21	12.6	34.88	3.5	33.75
Total	100.0		100.0		100.0		100.0		100.0		100.0	
Pct ash, as received	25.12		26.54		24.56		23.48		23.05		21.06	
Rate of flow, gpm	9000		6300		3500		53		3447		673	
Pct solids, dry wt	14.5		7.0		25.6		53.1		25.3		44.0	
Average tons solids per hr	349.8		115.0		250.0		9.0		241.0		95.5	

* Represents tonnage and flow from breaker. Actual flow into 24-ft tank would include an additional 800 gpm in closed circuit carrying 15.2 tons per hr from points 12 and 17.

Sample Point on Flowsheet	7 Feed to No. 5 Buck Launder Screens		8 No. 5 Buck from 24-Mesh Launder Screens, Feed to 16-Ft Hydroclassifier		9 Underflow No. 5 Buck Launder Screens, Feed to 15-Ft Tank		10 Under- Flow 15-Ft Tank		11 Overflow of 15-Ft Tank, Feed to Recircu- lating Pump		12 Overflow of Wemco Spiral to 24-Ft Hydro- classifier	
Standard Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8											0.4	
10											2.2	7.49
14	0.2		0.5								3.2	7.63
20	4.2	20.41	6.6	20.85	0.4						3.2	8.35
28	19.3	25.74	24.9	22.59	0.7	25.41	Unable to sample				2.8	8.32
35	17.7	24.56	28.2	23.75	3.7				0.6		3.2	9.20
48	13.2	25.63	14.3	22.89	11.8	24.08			1.6	11.90	4.6	9.80
65	8.5	25.79	8.2	22.34	18.8	22.80			6.4	13.62	8.2	11.91
100	10.2	24.92	4.6	24.98	15.6	24.92			15.0	17.22	7.4	14.63
150	4.7	25.11	2.7	24.54	10.0	25.74			10.0	19.82	6.0	17.49
200	3.5	25.93	2.1	25.44	6.7	26.58			7.6	22.94	58.8	34.39
-200	18.5	35.45	7.9	34.65	32.3	36.52			58.8	35.04	100.0	
Total	100.0		100.0		100.0				100.0		100.0	
Pct ash, as received	25.73		23.58		28.42				27.52		23.98	
Rate of flow, gpm	2774		1240		1534		734		800*		400	
Pct solids, dry wt	19.0		32.2		7.1		15.1		6.6		8.5	
Average tons solids per hr	145.5		116.7		28.8				13.7		8.9	

* Part of this water is used at times on the launder screens for dilution.

Sample Point on Flowsheet	13 Feed to No. 4 Buck, Two 6-Ft Wilmut Hydro-tators		14 Refuse from Two 6-Ft Wilmut Hydro-tators		15 No. 4 Buck Coal from Two 6-Ft Wilmut Hydro-tators		16 Refuse from 16-Ft Wilmut Hydro-tator Classifier		17 Overflow 16-Ft Wilmut-Hydro- tator Classi- fier to 24-Ft Hydro- classifier		18 Two 6-In. Spigot Dis- charge from 16-Ft Hydro- tator to Dewatering Screens	
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8	1.7	19.53	4.2	49.49	1.5	7.49	0.1					
10	22.9	23.75	33.1	74.08	21.0	9.60	0.6	68.00				
14	30.7	23.16	35.2	80.92	30.8	11.81	1.8	50.84			0.1	
20	27.5	20.46	19.9	80.49	27.0	13.74	17.9	42.32			4.8	6.75
28	9.6	20.27	5.4	71.90	11.3	15.37	34.4	61.59			22.8	8.65
35	2.9	16.45	0.7	42.68	3.4	15.49	27.4	80.46			27.2	11.52
48	1.3	16.30	0.5	37.27	1.6	15.07	12.0	86.80	0.4		16.1	13.78
65	0.8	18.58	0.3		1.4	16.10	4.0	85.00	1.0	8.66	9.7	17.17
100	0.8	21.72	0.3		0.7	18.95	1.3	68.55	4.4		7.0	21.86
150	0.4	25.03	0.2	42.28	0.3		0.3		7.8	11.05	3.4	25.35
200	0.4	24.44	0.1		0.6	25.74	0.1	48.39	7.8	15.15	2.1	27.87
-200	1.0	32.51	0.1		0.4		0.1		78.6	35.15	6.8	34.12
Total	100.0		100.0		100.0		100.0		100.0		100.0	
Ash, as received	21.99		76.48		11.81		65.36		30.40		13.97	
Rate of flow, gpm									400			
Pct solids, dry wt	61.9		68.7		69.1		67.6		6.1		33.9	
Average tons solids per hr	86.6		13.6		73.0		21.6		6.3		88.8	

Table I is continued on following page

Table I (continued). Analysis of Samples Taken from Coaldale New Fine Coal Plant

Sample Point on Flowsheet	19 Underflow of Dewatering Screens 30 and 35- Mesh		20 No. 5 Buck Coal from Dewatering Shaker		21 Feed to 45-Ft Dorr Hydro- Separator		22 Overflow of 45-Ft Dorr Hydro- Separator to 180-Ft Diam Thickener		23 Underflow of 45-Ft Hydro- Separator, Feed to Two 8x8-Ft Classifying Conditioners		24 Diluted Flota- tion to Two 6-Cell Banks of Denver No. 30 Flotation Cells		
	Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8													
10										0.1			
14										0.4			
20										1.2	14.06		
28										2.4	16.70		
35										7.7	19.46		
48										16.1	21.91		
65										18.9	22.66		
100										22.5	23.69		
150										13.1	26.83		
200										6.8	33.07		
-200										10.8	37.54		
Total		100.0		100.0		100.0		100.0		100.0			
Pct ash, as received		24.48		11.03		26.57		29.19		23.30		23.30	
Rate of flow, gpm		450				7754		7154		600		1120	
Pct solids, dry wt		10.7		66.3		7.3		4.5		34.2		20.0	
Average tons solids per hr		12.8		76.0		145.4		82.8		62.6		62.6	
Sample Point on Flowsheet	25 Refuse from Two 6-Cell Banks of Denver No. 30 Flotation Cells		26 High Ash Coal from No. 6 Cell Re-treated in No. 1 Cell		27 Coal from No. 1 Cell		28 Flotation Coal from Denver Cells Feed to Robins Dewaterizers		29 Underflow of Robins De- waterizers, Feed to DorrClone		30 Overflow from DorrClone to 45-Ft Hydro- separator		
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	
+ 8													
10													
14													
20													
28													
35													
48													
65													
100													
150													
200													
-200													
Total		100.0		100.0		100.0		100.0		100.0		100.0	
Pct ash, as received		79.25		18.42		15.23		13.45		13.60		24.26	
Rate of flow, gpm		610						510		410		270	
Pct solids, dry wt		6.0		42.2		33.1		36.1		23.6		3.3	
Average tons solids per hr		10.1						52.5		26.3		2.5	
Sample Point on Flowsheet	31 Flotation Coal from Underflow of DorrClone		32 Flotation Coal De- watered by Robins Dewaterizers and DorrClones		33 No. 5 Buck and Flotation Coal Blend Loaded Into Railroad Cars		34* Overflow of 180-Ft Thickener Tank						
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct					
+ 8													
10													
14													
20													
28													
35													
48													
65													
100													
150													
200													
-200													
Total		100.0		100.0		100.0		100.0					
Pct ash, as received		12.36				11.93		11.48					
Rate of flow, gpm		140											
Pct solids, dry wt		52.4				63.9		65.35					
Average tons solids per hr		23.8				50.0		126.0					

* There are 5000 gpm of this clarified water recirculated to breaker; balance of clarified water discharges into creek.

Table II is an equipment list giving information as to make and size. It is believed this information used in conjunction with the flowsheet and operational data will prove of value in design or operation of fine coal plants.

Sizing, Hydraulic Classification and Screening

Requirements: It is readily understandable that screening of 9000 gal of slurry would involve considerable capital expenditure for mechanical screening equipment, together with continuing maintenance.

Table II. Equipment in Coaldale New Fine Coal Plant

No. 4 and No. 5 Buck Section of Plant	Manufacturer	No.	Size or Capacity	Pump Head, Ft	Motor Rating, HP	Remarks
Hydroclassifier tank		1	24x10 ft			Prefabricated and assembled in field
Feed pump to launder screens	Barrett, Haentjens & Co.	1	3500 gpm	52	75	
No. 4 and No. 5 buck launder screens	Lehigh Navigation Coal Co.	2	3 ft x 8 ft 9 in.			Oversize screen dressed with 4 mesh stainless steel
		3	3x23 ft			No. 4 buck screen dressed with 12 mesh stainless steel
		3	3x23 ft			No. 5 buck screen dressed with 24 mesh stainless steel
No. 4 buck hydrotators	Wilmot Engineering Co.	2	6-ft diam Two 1500 gpm Two 6x19 ft		Two 10 Two 15 Two 7½	Slate conveyor Pump (Barrett, Haentjens & Co.) Shaker Screen 6x9 ft of Prizer tread 3/64x1/16x¾ in. 6x9 ft of 3/64 in. round hole For dewatering No. 4 buck
60-in. spiral classifier triple pitch (pitch 4 in. in 12 in.)	Western Machinery Co.	1	5 ft 6 in. x 27 ft 9 in.		10	
No. 5 buck hydrotator classifier	Wilmot Engineering Co.	1	16 ft diam 2400 gpm	18	10 15 5 Two 7½	Slate conveyor Pump (Barrett, Haentjens & Co.) Agitator Shaker Stationary panel screen 30 mesh Main dewatering screen 35 mesh
Tank classifier		1	Two 3x4 ft Two 4 ft 4½ in. x 20 ft 1¼ in.			
Recirculating pump	Barrett, Haentjens & Co.	1	15x10 ft 1500 gpm	100	50	Prefabricated and assembled in field
Flotation Section of Plant						
Hydroseparator	Dorr Co.	1	45x10 ft		15	Collecting rake speed 1.33 rpm Hydraulic rake lift Adjustable stroke
Dorrco duplex diaphragm pumps	Dorr Co.	2	90 to 560 gpm		Two 7½	
Classifying conditioner	Denver Equipment Co.	2	8 ft diam		Two 10	Conical bottom tank with airlift
Flotation cells, No. 30 Sub-A	Denver Equipment Co.	12	100 cu ft per cell		Six 25 Four 1/3	Froth paddle drive motor
Double overflow cells						
Flotation cell coal pump	Barrett, Haentjens & Co.	1	500 gpm	76	20	
Flotation coal dewaterizers	Hewitt-Robins Inc.	2	4x16 ft		Two 10	4x12-ft section ¼ mm 4x4 ft Section ½ mm
Cyclone pump	Barrett, Haentjens & Co.	1	500 gpm	116	30	
DorrClone cyclones	Dorr Co.	2	12 in. diam			
Turbo compressor	Spencer Turbin Co.	1	1350 cfm		7½	Volume at 12-oz pressure
Reagent feeders	Clarkson Co.	8	Model E			
Reagent distributors	Dorr Co.	2			Two ¼	
*Density control	Foxboro Co.	1				For regulating 45-ft hydro. rakes
*Density control	Mine & Smelter Supply Co.	1				For regulating cell density
*Geary-Jennings sampler	Galigher Co.	3			Three ¼	For sampling feed, coal, and refuse

* Installation not completed.

nance cost. Also it was recognized that screening of finer sizes is a difficult problem and that hydraulic classification, which is more practical, should be utilized wherever possible.

Size specifications for fine coal permit a maximum and minimum percentage of oversize and undersize in final products; as a result there is some overlapping of size between coals. The undersize or -20 mesh in No. 4 buckwheat is restricted to approximately 20 pct, whereas No. 5 buckwheat has no lower size limit according to anthracite specifications. In accordance with the laws of classification, efficient cleaning of No. 4 buckwheat hydrotator feed would depend on removing +8 mesh oversize and maintaining -20 mesh as low as possible.

The No. 5 buckwheat at Coaldale is a combined product of coal made by two different processes, each dependent on the other for overall maximum efficiency. Important in this respect is the practical separation of the coarser fraction of the No. 5 buckwheat, i.e. -20 +48 mesh, to be cleaned by hydraulic classification in the 16-ft hydrotator classifier; also important is the classification of a -28 +200 mesh fraction that can be economically cleaned by froth flotation.

Preliminary Classification: The major portion of -35 mesh in the feed of 9000 gal per min flow is removed in a 24x10-ft high hydroclassifier. There are no collecting rakes in this classifier. The sizing is based on settling velocity and rate of flow. The recovered +35 mesh fraction in the underflow is pumped to screens by a 3500 gpm centrifugal pump. Because closed circuits exist with respect to the 24-ft hydroclassifier, i.e. sample points 12 and 17 of Table

I (the overflow of the Wemco spiral classifier and the overflow of the 16-ft hydrotator-classifier) the actual flow to the 24-ft hydroclassifier is 9800 gal per min carrying 365 tons per hr of solids. Operating results of this hydroclassifier are shown in Table III. To make a practical separation of the 35 mesh under turbulent conditions an overflow rate or an equivalent settling rate of 140 ft per hr is required.

Table III. Overflow Rate Data for 24-Ft Diam Hydroclassifier

	GPM	Cu Ft Per Hr
Feed*	9800	78,400
Underflow	3500	28,000
Overflow	6300	50,400
Net diameter tank**	21 ft to 3½ in.	
Net area tank	358.7	
Overflow rate, ft per hr†	140.5	

* 9000 gpm breaker feed + 800 gpm recirculated load = 9800 gpm.

** 23 ft 11½ in. - (2 ft x 1 ft 4 in. inside launder) = 21 ft 3½ net diam tank.

† Overflow rate, ft per hr = Overflow volume, cu ft per hr

Area of tank, sq ft

Sample 2 of Table I shows results of preliminary classification. The removal of 115 tons of fines in 6300 gal of overflow has direct bearing on the problem of screen design for sizing No. 4 and No. 5 buckwheat before cleaning.

Launder Screens: Before the remaining 250 tons per hr were screened, the efficiency of several types of screens was investigated. It is well known that with fine sizes, once water is removed screening stops, and only water added by sprays or flooding,

as in pools, will increase the efficiency. Use of both vibrating and reciprocating shakers in other operations verified this condition. Following Parton's study² of launder screens in operation at Nesquehoning preparation plant, this type of screen was installed at both the Lansford and Tamaqua plants, having proved more efficient in sizing smaller anthracite coals. Launder screens are stationary, with screen cloth fitted over a wooden grid divided into compartments, or cells, 6x6x6-in. along a launder. In each cell a hole is drilled for an insert bushing or orifice. Since sizing depends on maintenance of suitable hydraulic conditions, essential requirements for operation of launder screens may be listed as follows:

- 1—Solids in feed should not exceed 25 pct.
- 2—Slurry should be distributed evenly over the screen.
- 3—Flow over launder screens should be regulated by orifices of such diameter that essentially all the water will have been removed through the cloth by the time the flow reaches the discharge point.
- 4—Enough water should be allowed to overflow the end of the screen to provide continuous removal of the product.
- 5—The slope of the screen, which may or may not be fixed, must maintain sufficient velocity to transport the product to the discharge point.

Table IV contains more specific data relating to launder screens. Operating results of launder screens are revealed by a study of sample points 3 to 9 inclusive, Table I. Fig. 2 is a view of launder screens in operation.

In screening No. 5 buckwheat it is important to note that considerably more fines and a higher dilution report in the product from the screen, see sample point 8 of Table I. This was purposely done to provide the required amount of make-up water for operation of the 16-ft hydrotator classifier.

No. 4 Buckwheat Feed

Dewatering and Classification: It is necessary to provide a dewatering step before feeding to the No. 4 buckwheat cleaning equipment. A 60-in. Wemco spiral classifier, shown in Fig. 3, was installed for this purpose. This method of dewatering has an added advantage of removing by classification the undersize carried with water from launder screens. Sample points 6, 12, and 13 of Table I illustrate performance of this unit. From a feed of 95.5 tons per hr, 8.9 tons of undersize are removed in a flow of 400 gal per min with little loss of +20 mesh. A closed circuit with this overflow has eliminated direct losses.

A change in size characteristics of feed on the second shift made it necessary to install an adjustable overflow weir mechanism, by which the pool area could be decreased from an effective area of

66 sq ft to 44 sq ft. This change in pool area resulted in reduction of the excess —20 mesh by classification and permitted use of the same screens required for day shift operation.

The Wemco spiral has proved an efficient machine during surge periods. Normal feed rate to this unit is approximately 90 tons per hr, but it has successfully handled twice this tonnage with no delay in operating time. It is also more efficient in dewatering and removing undersize than is the conventional tank with drag conveyor used at Lansford and Tamaqua fine coal plants.

Classification of Flotation Feed: Classification results of the 75-ft diam hydroclassifier at Tamaqua, operating under conditions similar to Coaldale requirements, were studied. This classifier, having an equivalent settling rate of 15.9 ft per hr, indicated that classification at 200 mesh was being made with high efficiency.

For the Coaldale operation it was decided to make a separation at 100 mesh. An investigation was conducted to determine the size of hydroseparator to give the most practical classification at this mesh. Fig. 4 shows results of laboratory long tube tests to determine percentage of —100 mesh in the total solids that settled and could be considered underflow product, together with the percentage of +100 mesh in the total solids that were siphoned off and could be considered overflow product.

The curves developed do not indicate any sharp break in classification, and to provide for minimum loss of +100 mesh in the overflow, an arbitrary free settling rate of about 40 ft per hr was selected. At this settling rate 33 pct of —100 mesh reported in the underflow. An attempt was made to confirm this figure by a pilot test using a 12¾-in. diam cone classifier. It was necessary in this test to feed the cone a representative sample from the 9000 gal in the preparation plant discharge flume. Results as shown in Fig. 5 and in Table V represent conditions when the test was run having an equivalent settling rate of 37.3 ft per hr. The +100 mesh loss in the overflow was slightly higher than loss from the long tube test, owing to more turbulent conditions in the cone. The percentage of —100 mesh in the underflow checked reasonably well with the long tube test.

Laboratory tests together with pilot tests indicated that a 45-ft diam hydroseparator should be used, having an inside launder to give a 42-ft 9-in. net diam tank and an equivalent settling rate of 40 ft per hr.

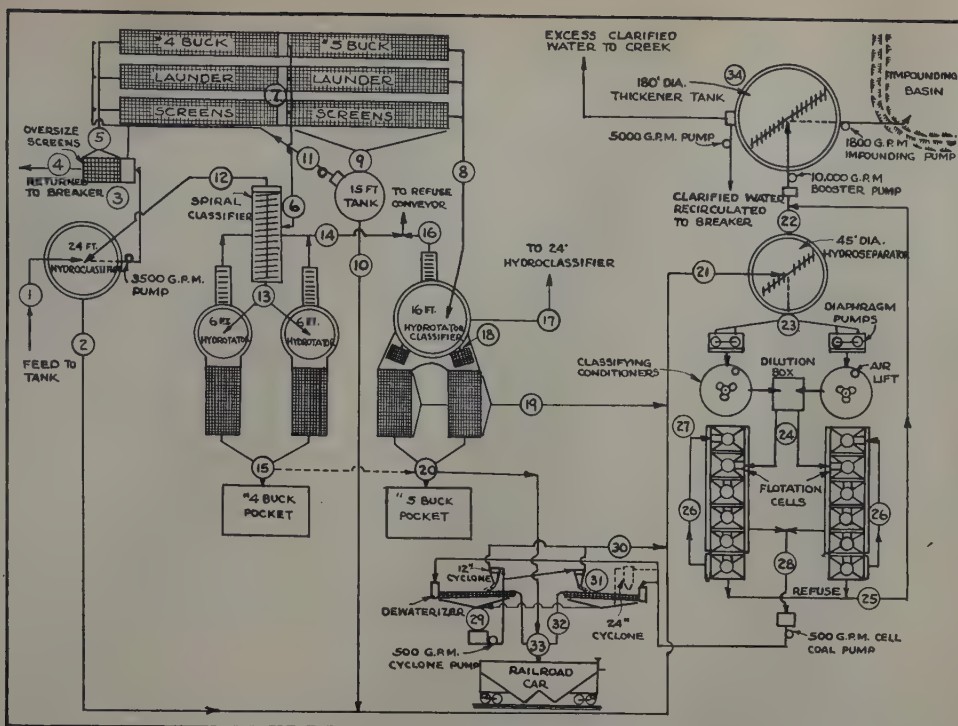
For comparison to the pilot test, actual operating results of the 45-ft hydroseparator having an equivalent settling rate of 40.7 ft per hr are plotted in Fig. 5. Because there was more oversize in the feed in pilot plant tests, results show a slight variation in the coarse sizes. This was to be expected prior to construction of the new plant for No. 4 and No. 5 buckwheat.

Table IV. Launder Screen Data

Launder Screen	Length, Ft	Width, Ft	Total Area	Slope Per Ft	Orifices Per Screen	Size of Orifice, ID	Stainless Steel Screen Cloth Mesh	Wire Diam	Opening In.	Open Area, Pct
Two oversize	8 ft 9 in.	3	52.5	1½ in.	90	1¾ in.	4	0.047	0.203	65.9
Three No. 4 buck	23 ft ¾ in.	3	207.6	1½ in.	240	13/16 in.*	12	0.023	0.060	51.8
Three No. 5 buck	23 ft ¾ in.	3	207.6	1½ in.	240	13/16 in.*	24	0.015	0.0267	41.1

* ¾ in. orifice inserts are used to adjust screen discharge flow to the desired amount for removing solids off screens.

Fig. 1—Flowsheet of the Coaldale new fine coal plant. The circled numbers are sample points.



Theoretically there should be very little -200 mesh in a hydroseparator underflow having this overflow rate, but because of the volume of water required to pump the underflow, which is the same as the overflow water, a normal amount of approximately 10 pct -200 mesh can be expected in feed to flotation cells. The elimination of the greater percentage of -200 mesh in the final coal product is discussed under Dewatering.

Table V. Results of Pilot Classification Test

Item	12¾-In. Test Cone		45-Ft Diam Dorr Hydroseparator	
	GPM	Cu Ft Per Hr	GPM	Cu Ft Per Hr
Feed	4.45	35.6	7754	62,032
Underflow	0.35	2.8	600	4,800
Overflow	4.10	32.8	7154	57,232
Net diam of tank	12¾ in.		42 ft 4 in.*	
Net area of tank	0.88 sq ft		1407.3 sq ft	
Overflow rate, ft per hr**	37.3		40.7	

* 45 ft 0 in. — (2x1 ft 4 in. inside launder) = 42 ft 4 in. net diam of tank.

** Overflow rate, ft per hr = $\frac{\text{Volume of overflow, cu ft per hr}}{\text{Area of tank, sq ft}}$

Sample points 21, 22, and 23 indicate that from a feed of 145.4 tons per hr to the hydroseparator 62.6 tons per hr at 23.3 pct ash is recovered as feed to the flotation plant. The preparation plant waste water carrying the remaining 82.8 tons per hr is pumped to a 180-ft thickener for clarification. Sample point 34 shows that the small loss of solids in the overflow of the 180-ft thickener is all -200 mesh and well within the limitation of 1000 ppm allowed by anti-stream pollution laws. From the overflow of the 180-ft thickener 5000 gal per min are recirculated to the preparation plant. The underflow of this thickener is pumped to a waste impounding basin.

Coal Beneficiation

No. 4 and No. 5 Buckwheat: Beneficiation of No. 4 and No. 5 buckwheat is accomplished in machines employing the same fundamental principle of separation, a separation based on a differential settling rate of particles in a relatively dense media maintained in suspension by an upward rising current.

The design of a Wilmot 6-ft hydrotator, as used in cleaning No. 4 buckwheat, necessitates recovery of fine solids in the water which overflows the separating tank with the cleaned coal. This being the media used for producing the required gravity in the separating tank, added water is eliminated in the recirculating sump overflow. Any build-up of these fines in the circulating load of media has the disadvantage of raising gravity in the separating tank, resulting in a higher ash coal product. For this reason considerable importance was attached to removal of fines by the Wemco Spiral Classifier. Inefficient sizing before cleaning necessitates continuous or intermittent removal of fines from the recirculating system.

Design features of the Wilmot 16-ft hydrotator classifier do not present the same problem. Overflow product of this machine, together with undersize, is removed by dewatering shakers and discharged directly from the circuit.

In the No. 4 machine the pump used for recirculating media and creating the upward rising current is outside the separating tank. In the No. 5



Fig. 2—Launder screens for separating No. 4 and No. 5 buckwheat.



Fig. 3—Two views of the Wemco 60-in. spiral for dewatering and removing undersize in No. 4 buckwheat before it enters cleaning equipment. Pool overflow and dewatered feed are shown.

machine the pump is inside the separating tank and recirculates a specific volume from the top classification zone. There being no closed circuit of fines to the system as in the No. 4 hydrotator, it was considered essential that water and solids required for maintaining the proper gravity classifying condition be obtained from the launder screens.

A comparison of sample points 13 and 8 illustrates feed conditions to the No. 4 and No. 5 machines. Whereas 86.6 tons per hr at 61 pct solids were fed to No. 4 hydrotators, 116.7 tons per hr at 32.2 pct solids were fed to the 16-ft hydrotator classifier. With the increased volume of water required for operation of the 16-ft hydrotator classifier, it is understandable that screening efficiency for No. 5 buckwheat is intentionally lower.

In operation of 6-ft hydrotator and 16-ft hydrotator classifier, effective separating gravities in the hindered settling zone are affected by upward rising velocities. Calculated velocities for the No. 4 and No. 5 machines are 427 and 150 ft per hr, respectively. Operating results of the No. 4 hydrotator may be observed by a study of sample points 13, 14, and 15. Fig. 6 and Table VI show float and sink analysis of feed to the No. 4 buckwheat hydrotators, together with calculated cleaning efficiency. An efficiency of 97.4 pct is dependent on the factors previously discussed.

In determining efficiency of the No. 5 buckwheat 16-ft hydrotator classifier, shown in Fig. 7 and Table VII, it is necessary to consider all the products of this machine, i.e. sample points 8, 16, 17, 19, and 20. Normally sample point 16 is considered as cleaner refuse, but since the separating tank overflow, 17, and the screen undersize, 19, are reject products from the hydrotator classifier they have been calculated in the actual refuse on a weight percent basis. Since these products are low in ash, 30.4 and 24.4 pct respectively, they lower the cleaner refuse from 65.36 pct ash to a calculated 47.01 pct. Calculated cleaning efficiency with this lower refuse ash is 78.5 pct. The lower efficiency may not be representative of other plant operations; also it does not represent a loss in the Coaldale flowsheet. Both overflow and screen undersize are fine high ash products, and their rejection permits production of a coarse low ash No. 5 for blending purposes. A

study of the flowsheet, Fig. 1, and operating results of the flotation circuit shows the feasibility of this type of circuit, especially when flotation is employed.

Flotation: There are at present two schools of thought pertaining to flotation of anthracite. The processes can be simplified by referring particularly

Table VI. Float-and-Sink Yield Analysis of Feed to No. 4 Buckwheat Hydrotators

Theoretical Recovery of No. 4 Buckwheat from Float-and-Sink Graph, Fig. 6			Pct
			86.5
Actual Recovery Based on Ash Formula			
Item	Sample Point No.	Ash, Pct	
Feed	13	= 21.99	
Coal	15	= 11.81	
Refuse	14	= 76.48	
Recovery = $\frac{\text{Refuse Ash} - \text{Feed Ash}}{\text{Refuse Ash} - \text{Coal Ash}} \times 100 =$			
$\frac{76.48 - 21.99}{76.48 - 11.81} \times 100 =$			
			84.3
Cleaning Efficiency = $\frac{\text{Actual Recovery}}{\text{Theoretical Recovery}} \times 100 =$			
$\frac{84.3}{86.5} \times 100 =$			
			97.4

to the physical character of flotation coal on the cell surface, i.e., matte or froth flotation. The processes may be classified as follows:

1—Matte flotation (Lasseter process), mechanical cell, depends on use of greater quantities of flotation reagents to create a bulk or mass suspension of solids on surface. This process employs mechanical agitation to keep solids in suspension but does not use the volume of air required for maintaining a froth.

2—Froth flotation, pneumatic cell, uses a minimum of flotation reagents and employs no mechanical agitation in the cell but requires a large volume of low pressure air to create froth and maintain solids in suspension.

3—Froth flotation, mechanical cell, uses a minimum of flotation reagents and employs mechanical agitation, with or without supercharged air, to create froth and maintain solids in suspension.

Flotation cells now in use which are representative of the above condition are the Denver sub A, Lasseter type; the Steffensen cell; and the Denver sub A. These flotation cells all have respective merits,

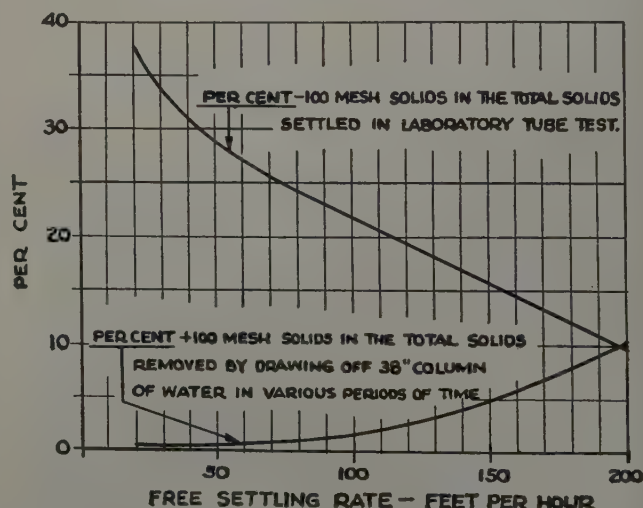


Fig. 4—Laboratory long tube test results for determining approximate amount of 100 mesh solids in settled and drawn-off product at various free settling rates.

particularly the Lasseter cell when it is necessary to float a coarser coal.

Matte flotation vs froth flotation of coal will no doubt be a controversial issue for several years to come. This paper will elaborate on the froth flotation process. Every consideration was given to creating what the writers believed to be fundamental requirements for froth flotation. Unlike metallic mineral flotation, whereby only a small weight percent of the feed is floated, froth flotation requires ample aeration to the pulp to float the mass. Important too are the size and number of bubbles and their dispersion.

Supercharged air at 8-oz. pressure is considered essential. An improvement has been accomplished by use of 16-bladed neoprene impellers instead of

Table VII. Float-and-Sink Yield Analysis of Feed to No. 5 Buckwheat Hydrotator

Theoretical Recovery No. 5 Buckwheat from Float-and-Sink Graph, Fig. 7				Pct
				82.9
Actual Recovery Based on Ash Formula				
Item		Sample Point No.	Ash, Pct	
Feed		8	=	23.58
Coal		20	=	11.03
Refuse	{	Cleaner Overflow*	17	= 30.40 × 0.1548
		Cleaner Refuse	16	= 65.36 × 0.5307
		Screen Underflow**	19	= 24.48 × 0.3145
				= 47.01 Pct Ash
Recovery = $\frac{\text{Refuse Ash} - \text{Feed Ash}}{\text{Refuse Ash} - \text{Coal Ash}} \times 100 =$				
$\frac{47.01 - 23.58}{47.01 - 11.03} \times 100 =$				
65.1				
Cleaning Efficiency = $\frac{\text{Actual Recovery}}{\text{Theoretical Recovery}} \times 100 =$				
$\frac{65.1}{82.9} \times 100 =$				
78.5				

* Portion of this is recovered by returning to 24 ft diam hydro-classifier.
** Portion of this is recovered by flotation by directing to 45 ft diam hydroseparator.

the 9-bladed impellers used at Tamaqua. The original test at Tamaqua with two 16-bladed impellers indicated that more lively small bubble froth could be expected, with resulting improved efficiency. Use of supercharged air together with a minimum +28 mesh in feed to flotation permitted lower quantities of reagents, consumption being 0.307 lb per ton feed of pine oil and 0.7 lb per ton feed of No. 2 fuel oil, as compared to Hagen's reported consumption³ in the Lasseter process of 0.25 lb per ton feed of an amyl ethyl alcohol frother and 5.0 lb per ton feed of kerosene.

Once coal reaches surface in froth flotation it is imperative that quick removal be made to prevent any serious showering of particles back into the pulp. Double overflows and where necessary four paddles instead of two as in the first two cells of each bank made this condition possible. A wetter froth was produced, with approximately 36 pct solids as compared to Tamaqua with single overflow at 46 pct. In this respect the Lasseter process offers a decided advantage in removing a coal product from the cells at about 48 pct solids. Use of double overflows makes effective the total area of cell. This is of fundamental importance in coal flotation. Efficient separation of +28 mesh particles in the feed eliminates any necessity for utilizing matte flotation.

Fig. 8 shows graphically the size consist of a froth flotation feed, together with an ash distribution

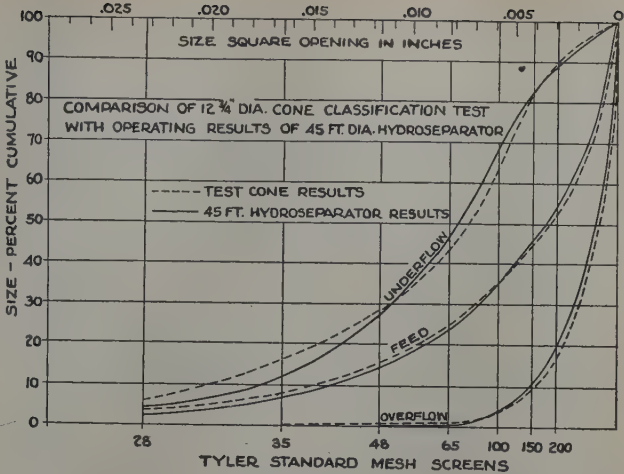


Fig. 5—Results of pilot classification test with 12¾-in. diam cone to determine size of hydroseparator compared with actual operating results of 45-ft diam hydroseparator. Statistics are shown in Table V.

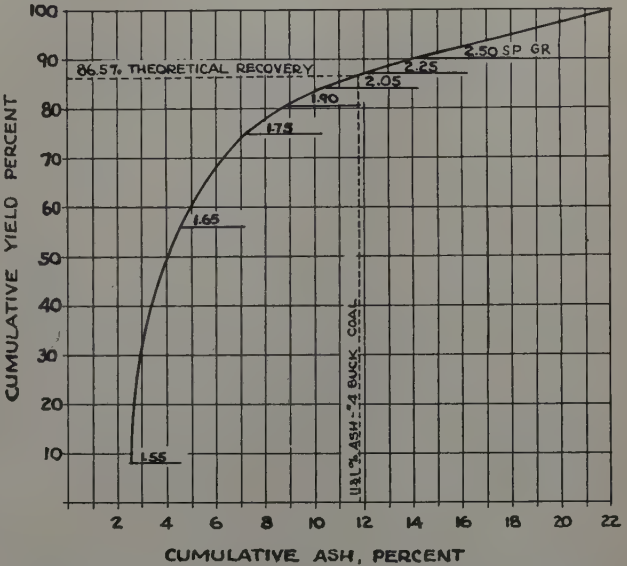


Fig. 6—Float and sink yield data of feed to No. 4 buckwheat hydrotators. Table VI gives accompanying statistics of recovery and cleaning efficiency.

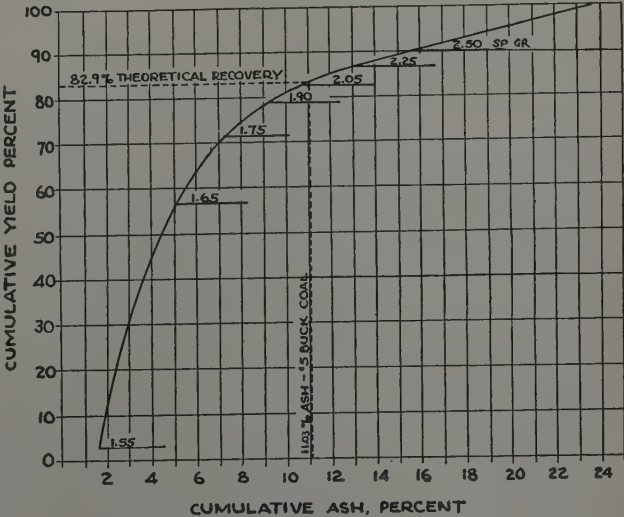


Fig. 7—Float and sink yield data of feed to No. 5 buck equipment. Accompanying statistics of recovery and cleaning efficiency are shown in Table VII.

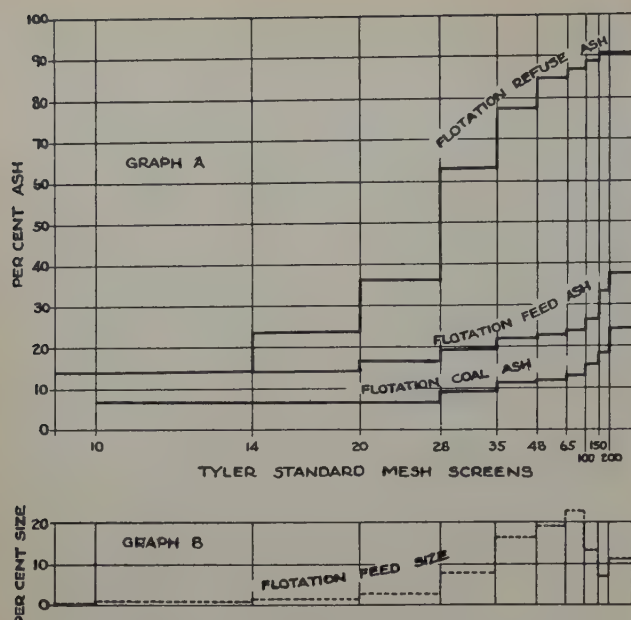


Fig. 8—Graph A, above, indicates ash distribution according to size consist of flotation feed, refuse, and feed. Graph B, below, indicates size consist of flotation feed.

analysis of sizes in feed, refuse, and coal. It is clearly shown that the particle size for efficient froth flotation cleaning should not be larger than 28 mesh.

In a paper describing Tamaqua flotation plant, Parton⁴ established that coarse coal required longer floating time and that it was necessary to re-thicken tailings of the primary circuit to maintain the required density of 20 to 25 pct solids to float the coarser coal. This may be better understood when it is realized that about 70 pct of the total coal floated is removed in the first two cells of the primary circuit, resulting in the tailings being reduced to about 10 pct solids.

It is interesting to note that in the Lasseter cell the grids of early Denver sub A cells have been replaced, a development which permits existence of an agitation zone, as well as a quiescent zone where there is less chance of the coarse particles being sheered from the bubbles by forces within the cell. Such a cell design may result in a more uniform flotation rate of all particle sizes.

The study of flotation at Tamaqua revealed that several important changes could be incorporated in the Coaldale flotation circuit to increase efficiency, provided several factors were taken into consideration, namely, elimination of coarse sizes in feed, a uniform feed rate, removal of coarse high ash particles in feed to cells, and the installation of automatic controls.

The flotation circuit treating a minimum of +28 mesh as developed for Coaldale eliminated the necessity of a secondary thickening operation. Two parallel banks of six cells are used. Feed enters cell 2, and cells 2 to 5 inclusive float a finished product. Cell 6 is used as a scavenger and by floating a high ash product it is possible to raise the ash of the tailings. Coal product from cell 6 is returned to cell 1, the cleaner cell. The froth product from cell 1 is a final coal product. Coal not recovered in cell 1 may be recovered, as the tailing is in closed circuit with the other cells.

In coal flotation plants every operator is repeatedly confronted with the variations in ash and size

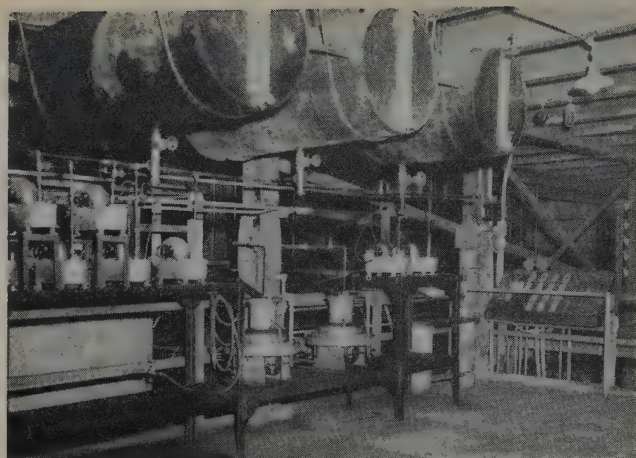


Fig. 9—Reagent feeder, distributor, and equipment control panel.

due to change in character of the feed. Operating conditions at Tamaqua and Coaldale indicated that during such a change feed tonnage may vary ± 50 pct and the ash from 20 to 30 pct.

Design and operation of the 45-ft hydroseparator allow 50-ton storage of feed. This has proved sufficient to take care of surges. Rakes in the hydroseparator have a 24-in. lift and are at present manually controlled by the flotation operator from the control panel. The setting of the rakes is dependent on density of conditioner feed at the discharge of the diaphragm pumps. Raising or lowering the rakes may be automatically controlled when the installation of a Foxborough density control is completed. Changes in rake setting are made when the operator finds a difference in gravity as measured by a pulp density balance. A gravity of 1.20 or 36 pct is maintained for conditioning.

To provide the operator with means of knowing the position of the rakes, company electrical engineers designed a signal system which is simple but invaluable. As the rakes are raised a plow attachment on the hydraulic lift pushes past a series of

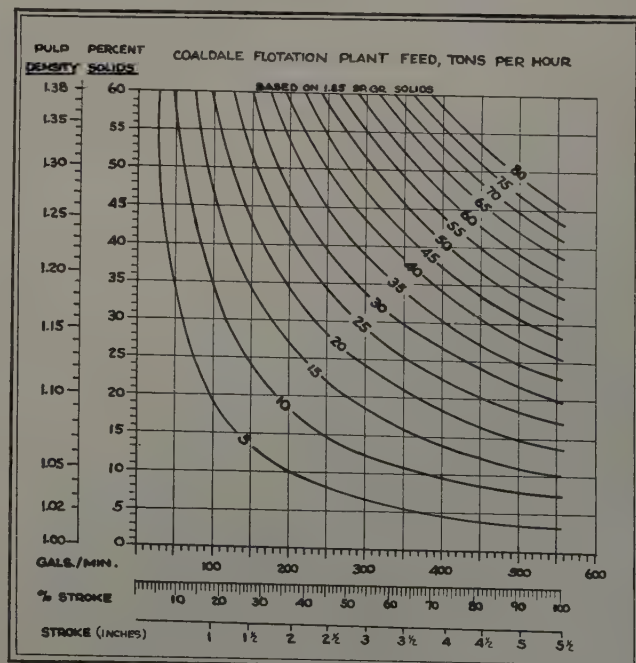


Fig. 10—Capacity of a single Dorr Co. duplex pump in handling flotation feed slurry at various pulp densities and pump strokes.

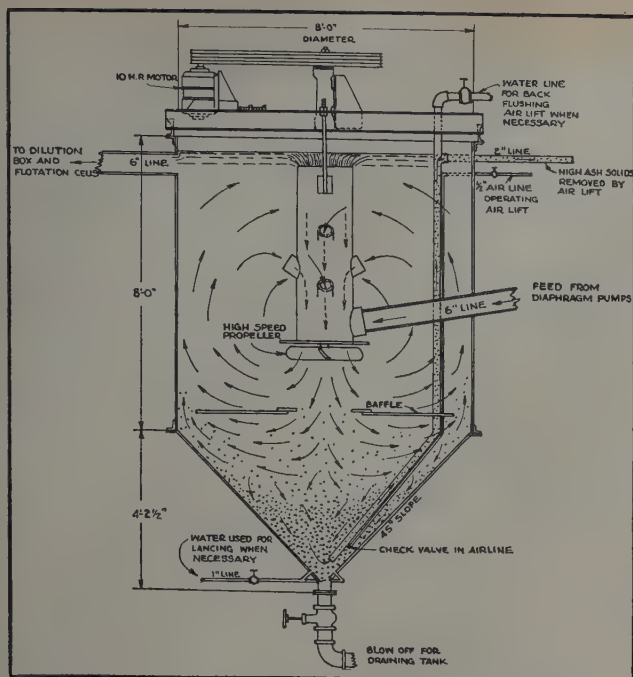


Fig. 11—Details of an 8x8 ft classifying conditioner with airlift.

five sensitive switches equally spaced and each connected to a light on the control panel set at the same spacing. These lights can be seen in Fig. 9, at the right of the photograph. During a period of surge the operator maintains a closer check on feed density and thereby establishes a most important control factor for efficient flotation.

In designing of the plant, provision was made for excess pumping capacity in the diaphragm pumps in case of an emergency. Normally each of these is operated at 300 gal per min rate of flow, but should the signal system indicate the rakes to be approaching the maximum lift, the stroke of the pump is increased until the surge is overcome. Fortunately this requirement is a rare occurrence, as flotation efficiency decreases owing to excessive feed rate.

Fig. 10 charts the capacity of a Dorr Co. duplex pump.

Classifying Conditioner

The classifying conditioner is a development which may prove a contribution to mineral engineering. Its operation at Coaldale colliery has greatly improved efficiency of coal flotation.

At Tamaqua plant it was customary practice regularly to blow off the 12x12-ft conditioner as often as once a shift to remove the accumulated load of pyrite and coarse high ash solids. This material, under severe conditions, accumulated to such extent that it overflowed with the feed to the cells. The Denver sub A flotation cell, although equipped with a sand relief to relieve cells of coarse particles, could not function properly; as a result the cells sanded up, and the froth became dead. With almost no flotation taking place the operator found himself in serious trouble and feed to the plant had to be by-passed to refuse.

A study of cells during such a period indicated that it was essential to maintain a condition in the cells which would permit proper dispersion of air. It was also observed that the accumulation of heavy solids in the cells was giving a short life to impellers, the neoprene rubber being nearly ground off the bottoms, the upper parts of the blades remain-

ing in good shape. A re-designed airlift installed in this conditioner eliminated this condition almost altogether.

Examination of solids that settled in the conditioner in the form of a cone showed that a very high ash fraction accumulated at the bottom. It was decided to investigate the designing of a conditioner that would incorporate a classifying action to separate coarse high ash particles and pyrite from the feed. The resulting machine is shown in Fig. 11. Essentially it is a Denver (Wallace type) super agitator with a classifying zone added. The fundamental principle involves a rising water current as created by the conditioner propeller. Since the propeller is designed to drive the pulp to the bottom of the conditioner, it may be observed that a baffle, having a circular opening in the center and a clearance between the side of the tank, now permits the pulp to drive down into a conical section and then flow in an upward direction through the outer clearance. With such a flow pattern it is necessary only to apply fundamentals of classification to make the necessary separation. To obtain the required rising velocity the hole in the baffle was made 2 ft in diam and so machined that varying size orifice openings could be used. Experiment to date has indicated that with a propeller speed of 240 rpm, an 18-in. orifice opening, and a 6-in. side wall clearance, an efficient separation of the objectionable impurities from the feed is made. A 45° angle cone was selected as a suitable bottom for collecting these impurities. A simple airlift is employed to remove solids sent to waste.

Results shown in Table VIII from an actual test in handling difficult bank material speak for the performance of this new classifying conditioner. In this installation feed to cells discharges through a 6-in. pipe, but in other uses it may be an advantage to overflow the rim of the tank to a collecting launder. To simplify the changing of upward rising velocity a variable speed drive could be utilized. Also in cases where the quantity of solids to be removed from the bottom of the cone is considerable it may be advantageous to use a diaphragm pump to control rate of discharge and percent of solids.

To make the operation of this plant more efficient the Foxborough density control previously mentioned and a Massco-Adams density control for maintaining uniform pulp density to cells are being installed. Galigher automatic sampling equipment



Fig. 12—General view of flotation cell floor.

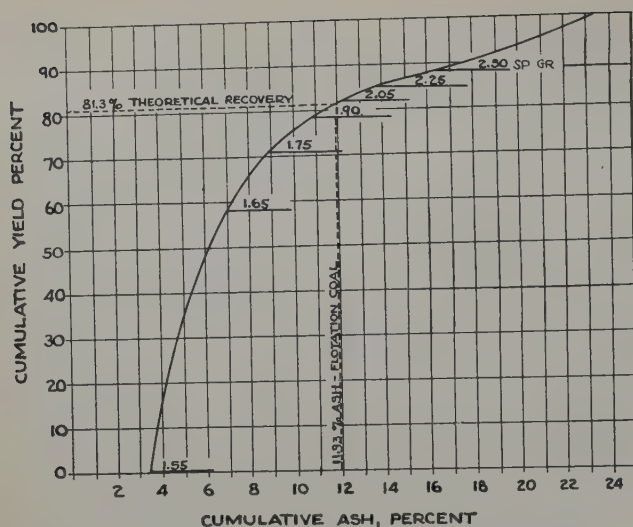


Fig. 13—Float and sink yield data of feed to flotation cells. Analysis of recovery and cleaning efficiency is given in Table X.

will replace hand methods for obtaining samples to represent daily plant performance.

Reagents

Fig. 9 shows the general layout of the reagent feeding equipment. Table IX represents consumption and distribution of reagents in the flotation circuit. Fig. 12 is a general view of the flotation floor with classifying conditioner in background.

Flotation Efficiency

Operating results of the flotation plant are best analyzed by referring to sample points 23 to 28, inclusive. Treating 62.6 tons per hr having an ash

Table VIII. Analyses of Classifying Conditioner Products from an Operation with High Ash Feed

Size Mesh	Feed to Conditioner		Feed to Cells Before Dilution		Airlift Refuse from Conditioner	
	Material, Pct	Ash, Pct	Material, Pct	Ash, Pct	Material, Pct	Ash, Pct
+ 10					0.0	
14	0.0		0.0		0.2	
20	0.3		0.2		0.8	36.28
28	1.5	26.91	1.3	13.16	3.9	56.80
35	12.2	34.08	10.7	20.60	22.6	69.02
48	22.5	37.01	21.4	26.94	30.5	68.47
65	22.6	41.47	22.8	33.04	21.2	73.02
100	23.6	41.96	23.7	39.93	14.2	65.67
150	10.0	45.96	11.3	45.89	4.1	64.96
200	4.5	47.86	5.1	50.49	1.5	63.65
-200	2.8	55.07	3.5	57.96	1.0	62.55
Total	100.0		100.0		100.0	
Composite ash		38.68		33.33		69.39
Solids, pct		33.6		34.4		36.5
Flow, gal per min.		600		540		60
Tons per hr		61.6		54.6		6.5

of 23.3 pct, a refuse of 79.25 pct ash and a coal of 13.45 pct ash is obtained. This is not considered the final flotation product, for in the dewatering system a 12-in. DorrClone, sample point 30, removes 2.5 tons per hr that is 89.0 pct -200 mesh with an ash of 24.26 pct. This is the high ash slime product mechanically entrapped in the froth. Sample point 32 shows the final product having an ash of 11.93 pct. The calculation for cleaning efficiency in Fig. 13 using the above analyses is 98.1 pct, see Table X.

The evidence supporting this flotation circuit to one incorporating a secondary thickening operation is the high ash refuse and the work being done by

the scavenger and cleaner cells which have froth products of 18.42 and 15.23 pct ash respectively.

It is not possible in the scope of this paper to discuss fundamentals of dewatering fine anthracite. Papers pertaining to this subject stress the difficulties encountered in this phase of the operation.^{5,6}

Primarily surface area is the major factor contributing to efficient dewatering. The more effective the elimination of -200 mesh, the smaller the surface area and the lower the resulting final moisture.

There is no particular problem in dewatering No. 4 and 5 buckwheat, the methods employed utilizing the conventional type of shakers. Sample points 15 and 20 indicate the lowest percentage of moisture from the shakers to be approximately 30 pct. Drainage in the car over a 24-hr period reduces this moisture content considerably.

Table IX. Consumption of Reagents

	Yarmor-F Fine Oil		No. 2 Fuel Oil		Reagent Distribution to Bank of Cells	
	CC Per Min	Pounds* Per Ton of Feed	CC Per Min	Pounds* Per Ton of Feed	Cell No.	Pct
					1 (Recleaner)	0
Two conditioners	44	0.087	200	0.350	2 (Feed)	50
Two 6-cell banks	112	0.220	200	0.350	3	15
Total	156	0.307	400	0.700	4	10
Gal per hr	2.47		6.34		5	10
					6	15

* Based on 62.6 tons of flotation feed, solids per hr.

Dewatering the flotation product presents an entirely different problem. It requires more elaborate equipment and it may include vacuum filtration, centrifugal filters, cyclones and screens. Of these the vacuum filter may be more efficient in lowering moisture content to 20 pct.

The Coaldale plant incorporated screens and cyclones to reduce the moisture content to approximately 33 pct so that it could be blended with the coarse No. 5 buckwheat having a similar moisture. Final drainage of blend product in ground storage or cars permits lowering of moisture to 12 to 14 pct.

The double overflow cells presented an unexpected problem in screen dewatering. It was not anticipated that the percent solids in the froth product would be decreased from 46 pct, as at Tamaqua with single overflow, to 36 pct with double over-



Fig. 14—Splashing and dewatering condition before installation of a 24-in. cyclone. Underflow feeding Robins dewaterizers.

flow. Increase in volume of water pumped on two screens dressed with ¼ mm wedge bar caused a flood condition, illustrated by Fig. 14. It has been necessary to employ the standby unit to dewater to 36 pct moisture.

Use of cyclones in dewatering a flotation product by screens is essential. Samples 28 to 32 show the results by this method. The feed in this circuit is pumped directly to the screen. About 50 pct of the solids having practically the same size consist as the

Table X. Float-and-Sink Yield Analysis of Feed to Flotation Cells

Theoretical Recovery Flotation from Float-and-Sink graph, Fig. 13				Pct
				81.3
Actual Recovery Based on Ash Formula				
		Sample Point No.	Ash, Pct	
	Feed	23	= 23.30	
	Coal	32	= 11.93	
Refuse	{ Cell Refuse	25	= 79.25 × 0.8016	68.34 Pct Ash
	{ Cyclone Effluent	30*	= 24.26 × 0.1984	
Recovery = $\frac{\text{Refuse Ash} - \text{Feed Ash}}{\text{Refuse Ash} - \text{Coal Ash}} \times 100 =$				
			$\frac{68.34 - 23.30}{68.34 - 11.93} \times 100 =$	$\frac{45.04}{56.41} \times 100 =$
				79.8
Cleaning Efficiency = $\frac{\text{Actual Recovery}}{\text{Theoretical Recovery}} \times 100 =$				
			$\frac{79.8}{81.3} \times 100 =$	98.1

* Portion of this is recovered by returning to 45-ft hydroseparator.

feed passes through the screen openings at 23 pct solids. This is pumped to two 12-in. DorrClones at 28-lb pressure, and the thickened products at 52 pct solids are discharged, for further dewatering, on to the coal layer formed from the feed.

This circuit has not proved satisfactory for dewatering or plant cleanliness. A circuit now in development stage will overcome the flooding condition as illustrated in Fig. 15 and permit the use of one screen. Flotation product is pumped at low pressure to a 24-in. cyclone having an extra deep cylindrical section. The value of this cylinder has not been definitely established but it is believed that with the high percentage of solids in the feed, classification can be made more efficient. The overflow of the 24-in. cyclone bypasses the screen and is pumped, together with the screen underflow, to one 12-in. cyclone at 28-lb pressure and discharged as in the original circuit.

The value of a cyclone circuit in dewatering flotation coal is in elimination of the -200 mesh. With

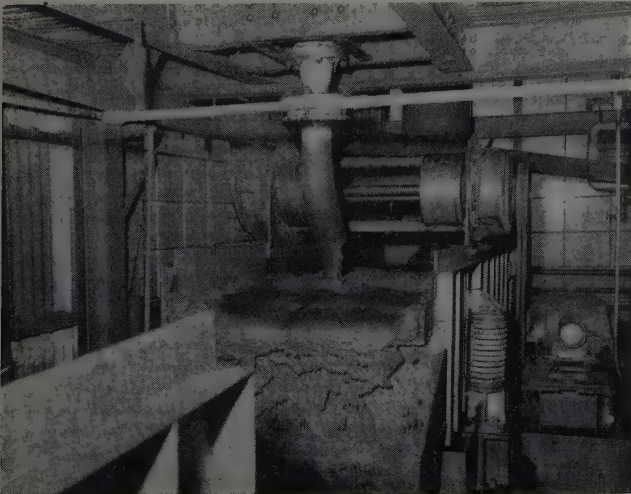


Fig. 15—Improved condition of dewatering with a 24-in. cyclone underflow feeding Robins dewaterizers.

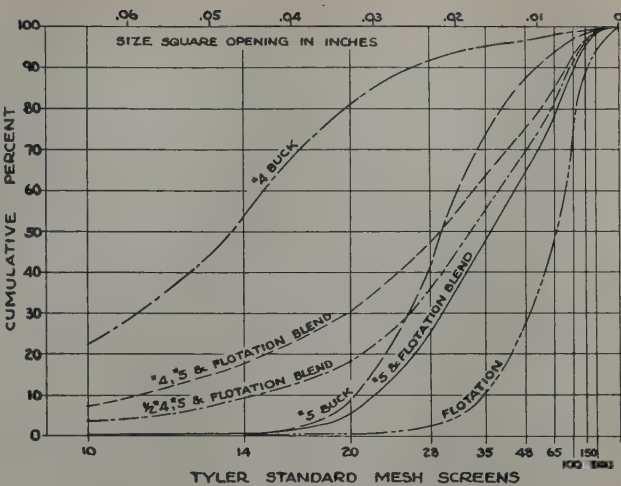


Fig. 16—Size consist of various coals produced from the new fine coal plant.

the secondary cyclone discharging to the 45-ft hydroseparator, any loss of coarse particles is recovered through the closed circuit.

Considerable study has been made of the effect of moisture resulting from unloading fine coal through hoppers of cars. Experiments in which extra water was added to the dewatered coal have proved that the greater the moisture content, the more compact the product and the more difficult its removal through the hopper.

Conclusion

In summary it may be stated that operation of this plant has more than justified the time required by the mechanical and preparation departments for its design and construction. The ash of the No. 4 and the No. 5 blended product ranges between 11 and 13 pct. Fig. 16 illustrates the size consist of the products which may be made for different market requirements.

Acknowledgments

The authors wish to thank the management for permission to present this paper and also the various members of the operating and engineering staffs of the Lehigh Navigation Coal Co. who designed and constructed this plant. The authors are especially indebted to the operators at Coaldale colliery for their efforts in maintaining high efficiency in the plant and to the staff in the chemical laboratory for the work done on the numerous samples submitted for analysis. Appreciation is also extended to the technical representatives of the many equipment manufacturers who were consulted during the development of the flowsheet.

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The Use of Wooden Rock Bolts in the Day Mines

by Rollin Farmin and Carville E. Sparks

TRIAL installations of rock bolts, of the slit-rod-and-wedge type, were under way at several units of Day Mines, Inc., when Korean hostilities interrupted the already slow deliveries of steel bars to the Coeur d'Alene district. Factory-made bolts had not yet been put on the local market, so the program was halted for lack of supplies. Interest was revived by a visitor's description of wooden roof bolts. These were said to have been used briefly with apparent success in a coal mine, until apprehension voiced by the U. S. Bureau of Mines caused the practice to be suspended.

To make wooden bolts for trial in ground support, Day Mines acquired a second-hand doweling machine equipped with two cutting heads, one to turn out the desired round rods of 2-in. diam, the other to turn out 1-in. rods to be used as powder-tamping sticks. This machine was installed in the all-weather sawmill of the Hercules mine unit at Burke, Idaho, where fabrication of the wooden bolts commenced early in 1951.

Most of the mining in the Coeur d'Alene district is along steeply dipping veins in shaly quartzite and argillite of Algonkian age. Ground support commonly is required in zones where the rocks have been sheared, brecciated, and hydrothermally altered. Pressure from the sidewalls is more troublesome than weight overhead, but both increase with the size of the mine opening. Caving may come from a progressive sloughing of irregular rock fragments or from an exfoliation and buckling of the layered wall rocks. The disintegration is thought to develop from an initial elastic expansion of the rock toward the newly-created mine opening, followed by the dilation of many tiny partings in the rock by absorption of water. As the partings widen, masses of rock develop weight and become free to fall. The function of rock bolts is to prevent or retard widening of partings in the rock supported.

Wooden Bolts, Wedges and Headboards

Bolt assembly used by Day Mines consists of a bolt 4 or 6 ft long, two wedges 16 in. long, and a headboard 30 in. long, Fig. 1. All four pieces are made of local red (Douglas) fir, either green or well-soaked in the mill pond before it enters the sawmill. Bolts are fabricated from cants, 2¼ in. sq, cut from relatively straight-grained timber with a minimum of knots and trimmed to 4- and 6-ft lengths. The bolt then is turned in the doweling machine from 2¼ in. sq to 2 in. diam round, except for a 4-in. length at one end which is left full square to provide the striking head and the shoulder that holds the headboard in position for wedging. The foot end of the bolt is slit with a thin saw for a length of about 16 in., thereby making a slot to receive the wedge against which the bolt is

driven for anchorage at the bottom of the rock hole. A similar slit, 12 in. long, is made in the opposite (head) end of the bolt to receive the second wedge, which crowds the headboard against the ground at the collar of the rock hole and puts the bolt in tension. The second slot is aligned 90° from the plane of the first slot to avoid longitudinal splitting and is notched out slightly to allow easier insertion of the collar wedge after the bolt has been driven to bottom. To prevent splitting the headboard by spreading action of the head wedge, this slot is oriented at 90° to the grain of the headboard when the pieces are assembled, Fig. 2. The wedges are similar to standard mine wedges, but more slender; they are cut 1⅞ in. wide and 1 in. thick at the heel and taper out in 16 in. of length.

The headboard, or bearing plate, is not necessary for some types of ground but generally is desirable because it helps the bolt to support an area of loose, friable rock and reduces the tendency for the rock at the collar of the hole to split away from the wedged head by distributing the pressure over a wider rock surface. The headboard may be a 24- to 30-in. length of 3-in. plank, 8 to 12 in. wide, but a similar length of rounded sawmill slab serves equally well at 20 pct of the cost. A hole of 2-in. diam is bored or punched through the center of the headboard, either at 90° or at various high angles to its surface. The bolt is inserted to its shoulder through this hole, then driven into the rock hole.

Bolts, wedges, and headboards are given a full timber preservative treatment to inhibit rot. Bundles of each are immersed in a warm saturated solution of Osmose salts in water for 48 hr, removed, dripped dry, and stored in a relatively humid underground depot to cure.

The current cost of fabricating a wooden bolt assembly follows:

Item	Untreated	Treated
Bolt, 1	\$0.13	\$0.17
Wedges, 2	0.064	0.08
Slab headboard, 1	0.09	0.18
	\$0.284	\$0.43

Most wooden rock bolts used by Day Mines are 4 ft long. Holes to receive them, about 42 in. deep and 2½ in. in diam, are drilled into the rock to be supported, nearly normal to the periphery of the mine opening. The type of drill used is dictated by convenience: stoper, jackleg, or jumbo-mounted drifter. Correct depth of the hole is assured by use of a measuring stick that has been cut to the proper distance from drill chuck to the ground at the collar of the hole when a standard length drill rod is at the bottom.

The bolt is seated to the shoulder through the hole in the headboard, the foot-end wedge is placed in its slot, and the assembly is inserted into the rock hole. Then the bolt is driven until it is seated solidly on the wedge against the bottom of the rock hole. Driving may be by hand with a sledge, or

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Discussion on this paper, TP 3555A, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, Nov. 24, 1952. Los Angeles Meeting, February 1953.

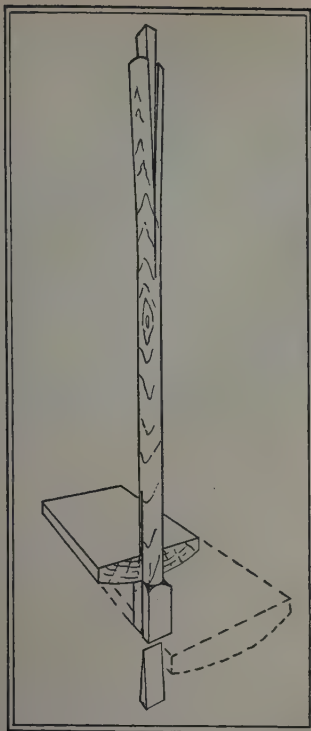


Fig. 1 (Left)—Drawing of a wooden bolt assembly used in the Day mines. Fig. 2 (Right)—Completed installation of bolts and slab headboards in the Dayrock mines. Heads of bolts have been trimmed off, and voids behind headboards have been filled with wedge butts and drill cuttings.

better still with a rock drill plus a rod and driving adapter, Fig. 3. A serviceable adapter may be made by welding a 2-in. length of 3-in. diam pipe to one side of a disc of 1-in. thick steel plate and to the other side welding the front half of a worn-out stoper chuck-bushing. In use, the ring of pipe cups over the head of the bolt and holds the driver in position despite rotation; the bushing on the opposite side of the plate holds the end of the driving rod against the plate to transmit blows from the machine.

The outer wedge is driven into the slot in the head of the bolt with an axe; considerably less force is required here than for seating the bolt on the bottom wedge. After a group of bolts has been installed, the job is completed by sawing off the protruding heads and wedges, close to the headboards, and painting the new tips with a timber preservative. The trimmings are used for filling in voids behind the headboards, together with sand, drill cuttings, or excelsior.

The trial installations of 1951 encompassed many of the types of application where steel bolts have been used: in drifts, crosscuts, stopes, raises, and sumps. These installations were intact a year later except where an inadequate pattern of bolting was used. Timber rot had started only in untreated headboards. The wooden bolt was pronounced a sound device, ready for expanded use. A new level was opened at the bottom of the Dayrock mine early in 1952, and the primary type of ground support has been the wooden bolt and headboard. Fig. 4 illustrates the changed shape of excavation for a crosscut where a conventional timbered opening, 10x10 ft, is converted to a 7x7½-ft bolted arch. The cost of the bolted crosscut is only 53 pct of the cost of the larger timbered crosscut.

Bolt spacing is dictated by the character of the ground and the nature of the opening to be preserved. For example, in the brecciated ground of

the Dayrock mine a sequence of seven bolt rings, spaced at 42-in. intervals along the length of the drift, Fig. 5, generally gives adequate support. Elsewhere, the same type of opening may require only a five-bolt ring or merely a single row of bolts along the hanging wall. Assorted patterns are employed in cut-and-fill stopes to restrain overhanging brows of waste rock while the ore is being removed and the fill placed.

Bolts with headboards may be damaged by blasting if they are installed too close to the working face; without headboards they may be placed nearby to reduce overbreak. Thus far, bolts with individual headboards have proved better than long boards pinned with several bolts, at least they are more easily fitted to irregular ground.

In view of the general success of the 2-in. bolt, experiments are being made currently with wooden bolts of smaller diameter. The 1⅞-in. bolt in a 2-in. rock hole appears to be adequate; perhaps



Fig. 3—Driving a wooden rock bolt.



Fig. 4—Looking from the new bolted crosscut toward the former timbered one, which required 47 pct more excavation.

slightly smaller ones may serve under proper circumstances.

Recently, the use of wooden roof bolts has been resumed in a coal mine¹ and an investigation by laboratory testing with scale models² reports a relatively high performance of wooden bolts compared with steel.

Applicability of Wooden Bolts vs Steel

The experience of Day Mines has led to the following opinions as to the merits of wood vs steel rock bolts.

Economics. The cost of the treated wooden bolt assembly of four pieces is about \$0.45 as compared with about \$1.50 for similar coverage in steel. Direct labor cost for installing the wooden assembly is about \$0.50 as compared with \$1.00 to \$2.50 for steel under similar circumstances. The investment in a stock of wooden bolts is only one-quarter that required for steel, even less with periodic local fabrication. Special tools are not needed for installing wooden bolts. An impact wrench is needed for the steel bolts, and in some cases different bits, rods, connections, and drills are brought in for the installation; where scattered working places are involved, a considerable investment in extra equipment is required. In very hard rock the larger hole to be drilled for the wooden bolt is a disadvantage favoring the use of steel bolts.

Availability. Wood is always available in ample quantity and is easily shaped with a minimum of shop equipment. Steel, on the contrary, becomes scarce-to-unavailable during wars or strikes in the steel, coal, and transportation industries.

Flexibility for Installation. The wooden bolt may be shortened readily with a saw to fit the length of a hole from which the collar ground has fallen away. A too-deep hole may be used by first inserting a filler length sawed from a spare bolt. Where the outer half of a wooden bolt has been blasted away by renewed excavation, the bottom half generally remains firmly anchored and may be put back into service when the protruding end is split and re-wedged. Steel bolts, on the other hand, need holes of the proper depth, and once anchored are rendered impotent if a foot or two of collar ground is blasted away. Steel wedges, angle washers, and bearing plates which fall into the ore stream become a mechanical hazard in the crusher plant, especially if the presence of magnetic minerals in the ore makes use of magnetic protection impractical.

Performance. Good anchorage is obtained by



Fig. 5—Soft ground supported by a series of seven-bolt rings. The broken rock in foreground fell during the installation.

wooden bolts in ground too soft for the wedged steel bolt, perhaps because the wood more completely fills the hole and prevents deformation of the soft rock into unfilled hole. The wooden bolt is more or less susceptible to timber rot but is free from attack by corrosive mine water, which may rapidly attack the threads of a steel bolt. Steel is better for building up a composite structure of bolts and channels because of its much greater strength in shear and bending. It also provides a better hanger for pipes and cables, unless the ground is too soft for good anchorage, and may be installed closer to the blasting face without damage than wood.

Safety. Hazards present during installation of wooden bolts are fewer than with steel. The miner uses only the familiar rock drill, whereas with steel he also lifts and operates the heavy impact wrench in an awkward overhead position, standing under ground not yet supported. The wedge which tightens the collar of the wooden bolt is driven from an offset position, after the bolt and headboard have provided partial support for the ground. In tension, 2-in. diam fir is somewhat stronger than 1-in. diam mild steel.³ Either is competent for the purpose. The relative weakness of wood to shear and bending strains is not usually involved unless wooden bolts are placed in interlocking structures or are subjected to blasting. The hazard of fire among installed wooden bolts does not appear great because of the spacing. On the other hand, steel bolt installations frequently invite electric welding and flame cutting underground.

Acknowledgments

The writers are indebted to Mr. Henry L. Day, President of Day Mines, Inc., for encouragement in the project and for permission to publish data. Many members of the staff and mining crews have contributed to the developments here described.

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Changing Factors in Mine Valuation

by Samuel H. Dolbear

THE value of a mine is basically dependent on its capacity to yield profits. Since the ore must be mined, treated, and sold, some of it in various future years, there is a risk involved as to future costs, selling price, and working conditions. It cannot be expected that the economic condition existing at the time of valuation will continue unchanged for long periods in the future.

During the past 20 years, mineral production in the United States has been conducted under a changing economy in many respects more exacting than that applied to other businesses. There have been increased production incentives, technical aid, exploration of privately owned mineral deposits by government at federal expense, and liberal loans for development and equipment, with risk partially assumed by government. Some of these benefits have been counterbalanced by price ceilings, consumption controls, and stimulation of competition from foreign producers who have been offered the same advantages extended to American operators.

For the present, mines will operate under a government policy directed toward reducing federal aid and control. The tenure of this change will depend upon future elections and the status of foreign relations. War and threat of war are now of the most vital significance to the mineral industries. Other factors which influence cost of production, markets, and price of mine output might be classified as Acts of God or Acts of Government. In some countries expropriation and the difficulty of exporting earnings or investment returns are risks that must be considered by foreign capital. Recognizing that this retards American investment in foreign countries, the Mutual Security Agency offers insurance against such expropriation and guarantees the convertibility of capital and profits.

Since it is impossible to predict with certainty either cost of production or selling prices of metals for long periods, some assumptions must be made as to profits in the future. The basic assumption must be that the price of the company's product will vary in proportion to changes in operating cost. There is often a lag in this reaction, however, for prices of minerals are generally more sensitive to declines and less sensitive to increases than are costs. This reflects in part the resistance of labor to downward wage revision and a corresponding alertness in realizing its share of price advances. Some labor contracts include automatic adjustments to metal prices.

Notwithstanding the complexity of the problems involved and the difficulty of weighing their effect on value, such risks may be appraised with reasonable accuracy and a rate of earnings adopted that is compatible with the risk.

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Discussion on this paper, TP 3599K, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, Nov. 30, 1952. Revised May 19, 1953. Los Angeles Meeting, February 1953.

It is, of course, possible to revert to a yardstick of value such as the *commodity dollar*, which has been advocated from time to time, but while revaluation in 1933 disturbed public confidence, the theoretical gold dollar continues to be the standard of greatest stability. Its gain or loss in purchasing power is reflected ultimately in cost of production and selling price of the mine product. At present 35 dollars are allocated to one ounce of gold.

Measurement of Risk

In the application of the Hoskold and most other formulae, a yearly dividend rate commensurate with the risk involved is set aside out of annual earnings. If the risk is great, this rate may be 15 to 25 pct of the amount invested. The remainder is placed in a sinking fund invested in *safe* securities such as high grade bonds or conservative equities, and the interest or dividends from these securities are added to the sinking fund. The sum of these sinking fund payments and the compounded interest at the end of the mine life is taken as the value of the mine.

Admittedly the decision as to the size of the risk rate is the most difficult element in valuation and one requiring the most exacting consideration. It is necessary to look years ahead in an effort to determine future costs, market prices, demand, competition which may develop, including that of substitutes, and other influences common to the mine and to the region in which it is situated.

Another phase of risk is the enactment of unfavorable legislation, taxes, and what appears to be an alarming spread of nationalization and expropriation. Capital is sometimes *borrowed* from the government to finance strategic production. Such loans may be collectable only out of production and involve no liability otherwise. Valuation in these cases must recognize the effect of such a reduction in liability.

Offsetting some of these risks are the possibilities of mechanization and other cost-reducing discoveries, improvements in mining and treatment methods, new uses for minerals and metals, and normal growth of markets.

In this paper, the terms *risk rate*, *dividend rate*, and *speculative rate* are synonymous. *Safe rate* and *redemption rate* are also used interchangeably. These alternatives are used here because they are commonly found in the literature on mine valuation.

In Michigan, the State Tax Commission has long employed a risk rate of 6 pct in its valuation of iron mines. There the outline of reserves is well established and operating costs and conditions are based on adequate experience.

The following comment on rates appears in the report of the Minnesota Interior Commission on Iron Ore Taxation submitted to the Minnesota Legislature of 1941.¹

Most engineers agree that 7 percent for the speculative rate is "an absolute minimum". C. K. Leith in

"Mineral Valuations of the Future" writes as follows:

"In practice a very wide range of interest rates is used, depending on the special local conditions, or upon the practice of particular appraisers. In general, the range has been between 6 and 25 percent, with the trend toward a figure of 8 to 10 percent in recent years for standard mining enterprises in which the factors are well known. Where interest rates as low as 5 or 6 percent are used, there usually have been off-setting corrections or discounts in the other factors in the valuation, such as tonnage, production or costs. In other words, part of the hazard is taken care of there rather than by the general interest rate."

The last two sentences in this quotation indicate the method adopted by the Michigan State Tax Commission in the appraisal of mines in Michigan for the purpose of determining values upon which real estate taxes are to be levied. F. G. Pardee, Mining Engineer and Appraiser of Mines, states that the plan used is to consider separately all the speculative features of the particular mine in direct proportion to their bearing on the valuation and their elimination as far as possible, before the interest rate enters the computations. This method of eliminating the speculative features justifies the use of 6 percent by the Michigan Tax Commission.

The rate commonly used for redemption of capital is 4 percent. It is true that the net interest rate or rate for absolute safety and stability of return at the present time is considerably less than 4 percent. When conditions again become normal the net interest rate will probably assume its former value and the assumed rate of interest should reflect normal conditions inasmuch as it is not selected for the immediate future but for the life of the mining property.

Manufacturing operations commonly employ a 6 pct risk rate. Mineral industries that are essentially manufacturing operations with mining or quarrying as secondary activities may also be justified in using a low figure. Some cement plants are examples of this class.

Under the Hoskold plan of valuation, risk rate is treated as a rate of annual dividends having first claim on profits. After payment of this dividend, the remainder of profits goes into a sinking fund which is, theoretically at least, invested in safe securities, formerly high grade bonds yielding 4 pct. The sum of this sinking fund accumulation (together with cumulative interest on the sinking fund) for the life of the operation is the value of the mine. *Present value* calculated by the formula shown under Non-Uniform Annual Income provides for time lapse between the date of estimate and the time when profit is actually realized from operations.

In recent years investment in safe securities yielding 4 pct has been difficult of realization. In 1947 Scudder, Stevens & Clark of New York surveyed funds of endowed colleges in the United States, in the amount of 1.3 billion dollars. The average yield was slightly under 4 pct. Life insurance companies realized somewhat under 3 pct and fire insurance concerns about 3.6 pct during the same period. Interest rates in 1951 to 1952 again established an upward trend. The average realization by endowed colleges in 1952 is believed to be about 4.5 pct. To some extent this is due to the increasing presence of common stocks in conservative portfolios. In valuation of short-lived operations the prevailing interest rates probably should be used. Where the life of the operation is of considerable length it is believed that 4 pct is a conservative figure. The distinction between short and long periods is one best left to judgment at the time of valuation.

Effect of Time on Value

Various segments of an orebody usually vary widely in assay value, and because of width, depth of mining, roof conditions, and other factors which change from year to year, the annual yield may vary widely.

A realistic valuation should be based upon a plan of attack in operations which separates the orebody into segments to be mined during uniform periods of time, usually a year, unless there is likely to be a radical variation in grade and cost at lesser time intervals. Valuation should, therefore, be based upon a plan of operation during which the ore to be mined each year is definitely allocated. The plan of operation must, of course, first determine at what point in the mine the stoping is to be commenced. If the valuation concerns an active operation, then the present location of existing faces is usually, although not always, the governing factor. It is possible, for example, that the value of the enterprise could be enhanced by a change of locus in early operations. The highest value is usually obtained by mining the highest grade ore first. This permits larger additions to sinking fund reserves in the early years of operation, and reduces the burden of interest on invested capital.

As Hoskold so clearly expresses it: "If money could not be employed and a marketable rate of interest obtained for its use, the value of any income or annuity would be equal to that paid at the end of one year, multiplied by the number of years the annuity has to run."

But since money can be loaned or invested to yield an income, the present worth of one dollar, payable at some future time, reflects the loss of such income, and it decreases with the length of the deferment period to the extent that it is almost valueless at the end of 40 years. Ore reserves are seldom valued beyond a 30 to 35-year period. If reserves are large enough to survive 35 years of operation, the residual reserve, i.e., that remaining after the 35 years, is worth, on the basis of Imlay actuarial tables, about 3¢ or less for each estimated profit of one dollar.

Residual Value

But it must be recognized that if the operation is revalued at the end of 35 years, or at some lesser interval, a new capital value is created, since the residual ore, or part of it, is brought within 35 years of realization. Thus ore which for this reason has negligible value today may have an established value one year from today.

It is therefore advisable to provide a statement of ore reserves expected to remain after 35 years, or at the end of the terminal date adopted, and the possibility of additional earnings in a period of post-terminal operations.

Uniform Annual Income

Occasionally mineral prices and costs of operation are of such a character that there is a substantial uniformity of income over a relatively short period, possibly two or three years. Many valuations have applied this supposition to longer periods by taking as bases what are considered averages of annual production value and costs. This is the basic assumption in the Hoskold premise. Such a procedure is believed to be faulty, for the presence of so many variables eliminates any possibility of uniformity over long periods. As pointed out elsewhere, higher grade parts of the orebody yielding maximum profits

in the early history of operation may produce a substantial increase in value as compared with the treatment of ore of an average grade during that period. Valuation based on the use of averages over the life of the mine is seldom if ever justified. The practice of maintaining a more or less uniform grade of ore in mill feed may at times produce some tax advantages, but it may be expected to yield a lower valuation.

Non-Uniform Annual Income

The use of formulae based on non-uniform income should be founded upon a careful plan of future production and an exhaustive weighing of all the factors that may influence financial results on a year-by-year basis for the life of the operation. This is especially important in a time such as the present, when results are dependent to a considerable degree on war or threat of war, tax legislation, and government controls and regulations, as well as prices and cost changes.

Often it may be desirable to develop two or more plans of operation, making spot checks on a specific year's results under each of the plans to determine procedure producing the optimum result. Calculation of profits should be made for each year separately, taking into account all predictable factors such as grade of ore to be mined, width of orebody, and roof conditions. Unpredictable costs, as stated, should be absorbed in the risk rate.

The Parks formula, developed to overcome this objection, has been discussed in detail elsewhere.² The formula applied to each year of unequal profit is as follows:

$$Vp = \frac{(\text{sum of } PmR^{n-m} \text{ series})}{1 + r' \frac{R^n - 1}{r}}$$

in which

- Vp = present value as of the date of the valuation
- P = yearly profit (may vary each year)
- R = \$1.00 + 1 year's interest at r rate
- n = life of operation, in years
- m = specific year in which income P is received
- r' = speculative (risk) rate
- r = safe rate

Depletion

The term *depletion* has been defined by E. S. Benson³ as follows:

Depletion from the tax viewpoint means the statutory deduction from gross income designed to permit the return to the taxpayer of the capital consumed in the production and sale of a natural resource. The mining enterprise realizes income on the extraction and sale of minerals and a portion of the income realized represents capital consumed. As the resource is exhausted, the mining enterprise approaches the end of its existence unless new sources of supply can be acquired. Depletion from the tax viewpoint is a creature of statute with limited meaning and application and, in essence, is a method for amortizing the value of the primary asset of a mining enterprise.

Determination of the amount of depletion allowable has become a complex matter, with changes taking place from time to time as new legislation is enacted and new rules are formulated by the Treasury Department. The present laws permit depletion on about 60 minerals and metals at rates ranging from 5 to 23 pct, to the extent that depletion may not exceed 50 pct of net earnings.

In the calculation of net earnings and their reduction to *present worth* by the Hoskold or Parks

formulae, deduction is made for estimated income taxes, and such deduction should not, of course, be applied to that portion of income arising from depletion. Depletion accumulations each year should be added to "net profit after taxes" in so far as valuation procedure is concerned.

Depreciation

The rate of depreciation used in financial records of mines is usually the maximum rate permitted under income tax regulations of the Bureau of Internal Revenue. These rates are based on the Bureau tables of average useful life ranging in various structures and equipment from 2 to 30 years. If the useful life is actually below these average figures, then the materials concerned may be non-existent before they are fully amortized.

In other cases material may be fully depreciated on the books, but the property may still have substantial remaining usefulness and therefore value. Under wartime certificates of necessity, plants have been amortized in periods as short as 5 years. This is permitted on the assumption that the plant would have little if any value except under emergency conditions. It is possible, however, that substantial value may remain at the end of so brief a period. The actual value, therefore, can be determined only by appraisal. Book value is an academic figure which might be far different from appraised value.

Markets and Competition

Many minerals have market characteristics that must be weighted in determination of risk. Coal is subject to special seasonal factors such as weather, to the control of output by labor in an extraordinary degree, and to the competition of other fuels. Iron and alloy metals are so closely related to military requirements that it is difficult to determine a normal market. The outlook for fertilizer minerals is based upon agricultural demand and is highly sensitive to farm income. Sales of fertilizer decrease promptly with a drop in farm income notwithstanding intensive educational work, which has offset some of that tendency.

Gold mining, with government limitation of price and no freedom of market; restriction in the use of supplies in emergency periods; and deterioration of mine and equipment in periods of idleness presents special problems in forecasting future value. Materials that are dependent on certain markets may suffer at times. When restrictions are imposed on building construction, for example, a long list of minerals and metals may be affected. On the other hand, war demand sometimes creates temporary prosperity for minerals not normally in demand at profitable levels.

Quicksilver, manganese, and chromite are examples of these in the United States, although generally speaking, in none of these are reserves found developed sufficiently for accurate valuation. And there are many marginal deposits of other materials, including staples such as base metals, that cannot operate on anything short of an emergency or war economy. Demand for building materials, cement, gypsum, sand, gravel, and others, is related to general prosperity and to cumulative demand for housing and engineering construction. Demand is reduced when scarce materials such as are used in electrical and plumbing fixtures are diverted into military channels. The influence of new demand based on inventions and engineering developments

is particularly important in this era of atomic progress and jet propulsion, when there is need for high-temperature-resistant materials. The prospect of large production of titanium metal is one example of demand based on technical progress. Expansion of production often means reduction of price, or perhaps this should be stated conversely, since the reduction of price may also be responsible for larger demand. The price history of aluminum is a case in point.

Competition, actual or potential, is an extremely important risk factor, and strangely, one that is even now too often ignored in expansion of mineral production. Uncontrolled increase in the number of potash mines in Germany, years ago, led to bankruptcy of dozens of producers and a reorganization of the industry under government control. The prosperous potash production in the United States could conceivably suffer a similar fate by overexpansion.

Captive mines usually enjoy more security of markets than independent operations. Coal mines operated as railway subsidiaries, iron mines controlled by steel mills, and asbestos production going into the manufacturing operations of a parent company are examples of this sort.

The threat of the development of substitutes is always present. Unless there is specific evidence of danger from this source it can usually be disregarded, for expanding markets are usually sufficient to offset loss by substitution. The widespread use of plastics in place of metal has apparently had little visible effect on demand. Just what the present position of metals would be in the absence of plastics might of course be considered. But, as stated, specific evidence of possible substitutes should be investigated, as for example, the proposal to replace much of the antimony in storage batteries with calcium.

The development of substitutes and their effective replacement of other materials requires considerable time, often many years. The change is usually gradual enough to permit absorption of the competition over substantial periods and need not therefore have any abrupt effect on value.

On a long range basis it is important to study the growth or decline in demand by examination of statistics over a period of years. The price trend should be treated in similar fashion. Graphs illustrating the history of production, consumption, and prices are extremely useful, and the reason for any anomalies in the pattern should be sought. Whatever the reason, statistics should be interpreted in terms of risk.

Valuation of Mining Shares

Procedure for the valuation of corporate shares differs from that used for a mining property. Under present tax regulations depletion allowance to a shareholder is limited to the purchase price paid for the stock. Presumably this is restricted to the amount paid by the original purchaser, and it is doubtful if a case can be made for a transferee. It cannot be distributed to the shareholder until earnings have first been fully paid out. Since tax legislation and regulations change from year to year, valuation must be based upon the legal situation at the time of valuation. The specific tax background is usually different for each corporation, and this makes an expert detailed analysis essential.

The value of shares is also affected by the dividend policy of the company. In most cases current

earnings are not completely distributed as dividends. That part of earnings carried into reserves comprises a residual value and its disposition may have a profound effect on the value of shares.

Dividend payments may not correspond to the risk rate used in mine valuation, and earnings carried into reserves may be more or less than the sinking fund payments in the mine valuation procedure. The value of shares is therefore dependent in a considerable degree upon the financial policy of the company, at least in so far as a minority shareholder is concerned. If the valuation is made on behalf of the holders of shares enough to control the company's disposition of earnings, then it may be desirable to make one valuation based on the current policy of the company, and another dependent on an adoption of changes in procedure that would yield the optimum results.

Periodic Revaluation

Revaluation made annually or at some other regular interval is in effect a periodic re-audit of the earning power and value of ore reserves. Annual audit of the financial position of an operation is of course generally accepted practice.

The ore deposit, although it is the most valuable asset of the company and may undergo radical changes in value from time to time, is nevertheless usually placed on the books at some pre-determined figure, and except for deductions for depletion, the original value remains unchanged.

Price increases may transform marginal into profitable ores and price decreases may erase parts of the orebody. Under these circumstances the estimated life of the mine and its value undergoes change. For example, when the price of gold was increased from \$20.65 to \$35 an ounce, gold mines were revalued to establish the profit position and property value at the new figure. Mines with by-product gold were also affected.

Failure to revalue mines may bring about conditions present in two large operations revalued by the writer. The book value of one mine had been reduced to about \$750,000 although earnings exceeded \$2,000,000 annually and ore reserves for many years' operations had been developed. The application of *discovery value* raised the book value to about 20 million dollars.

To cite another instance, the mine of an American-controlled foreign corporation had been valued by income tax authorities at \$750,000 in 1919. Since this was a highly profitable operation, depletion soon eliminated this figure, and when the mine was sold 20 years later for more than \$8,000,000 it was discovered that over-depletion amounted to around two million dollars. Tax authorities took the position that two million from over-depletion, plus the sales price, a total of \$10,000,000, was not a repayment of capital. They classified it as net earnings for the year in which the sale took place.

Periodic revaluation would of course have avoided the penalty for this oversight. Furthermore it provides information of importance to management and is of concern to shareholders.

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Block Caving at Bunker Hill Mine

by C. E. Schwab

A lead-zinc orebody, in fairly strong quartzite and with a dip of 35° to 60°, is block-caved by use of scrams in a stair-step pattern up the ore footwall. Scram linings to handle coarse muck and permit the use of folding scrapers are developed by the use of end-grain wooden blocks to reduce maintenance and keep operating cost to a minimum.

THE Bunker Hill mine, since its discovery in 1885, has steadily produced a high grade of lead-silver-zinc ore. By the end of 1952 over 21,000,000 tons of this high-grade ore had been produced by square-set mining, and reserves in the mine continue to be very satisfactory both as to quantity and grade. For many years prior to 1941, mine production and mill capacity had been 1200 tons of feed per day. Closely adjacent to the mill, and stored behind dikes, coarse jig tailings had been impounded during the time preceding the advent of fine grinding and selective flotation. When manpower became short in 1941 and sink-and-float preconcentration was proved successful, mill capacity was increased to 1800 tons per day to treat these jig tailings economically.

By 1946, because the supply of jig tailings was limited, underground exploration was started to discover and prove ore reserves of low-grade material which could be mined by an appropriate bulk mining method. During the years of square-set mining many possible areas of low-grade mineralization had been observed. One chosen for the first exploration work was sufficiently remote from active mining areas so that subsidence, if an orebody were proved, would cause no problem. Also, old adits and workings were still open and in good enough condition so that exploration in the mineralized zone could be started with a minimum of preparatory work.

In 1948 an orebody was proved of sufficient tonnage, of a grade about 2 pct Zn, 0.5 oz Ag, and 1.0 pct Pb. It was decided to use block-caving, the only appropriate mining method by which this grade of ore could be economically recovered. Exploration for additional reserves in other areas of the mine is continuing, but ultimate results are not known at this time. With more sink-and-float capacity, larger ball mills, and more flotation machines, mill capacity was increased to 3000 tons per day, permitting the mining of ore in the square-set area at a maximum rate not usually achieved, because of the scarcity of labor. Increased mill capacity also permits block caving and the mining of jig tailings at variable rates to keep mill feed up to 3000 tons per day.

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Discussion on this paper, TP 3621A, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, Feb. 27, 1953.

Fortunately the three types of feed are amenable to the same mill circuit and reagents for recovery of Pb and Zn. For example, during the first 10 months of 1952 square sets produced 827 tons per day, block-caving 1421 tons per day, and jig tailings 643 tons per day, an average daily production of 2891 tons for all three products.

Exploration had proved the existence of an orebody 1000 ft long and 165 ft wide in horizontal section, see Fig. 1. Company engineers were concerned only with the vertical extension, about 300 ft, from an old level to the surface. Much of this almost outcropped, Fig. 2. The ore lies in the hanging wall of a major fault of the Bunker Hill mine, standing at 65° in one end of the zone and separated from the fault by a wedge of waste, see Fig. 3. This wedge pinches out near the center of the zone, at which point the ore dips 45°, lying nearly on the fault, Fig. 4. The remaining portion lies on the fault and conforms to the fault dip of 35°, Fig. 5.

Open-pit mining for the top of the ore was considered, but since the ore zone dipped into and under the mountains, adverse waste-to-ore ratios precluded use of this method.

The ore occurs in massive quartzite of sufficient strength to support untimbered drifts, crosscuts, and raises. Zones of weakness in the quartzite are bedding, jointing, and small faults or slips. The mineralization, which occurs as small stringers of sphalerite and galena as well as pyrite, creates another line of weakness. The mineral veins or veinlets in themselves are high-grade. Their size and regularity and the amount of barren quartzite by which they are separated determined the limits of mineable ore, which are all *assay limits* except for the one determined by the major fault.

Block 1

Without any background of caving in this type of quartzite, engineers selected the first block on the very steep end of the zone. Compelling reasons prompted this decision. The steep portion of the ore in Block 1 was of the lowest grade, so that if difficulties were encountered no very valuable ore would be lost, while the experience gained might be applied in mining the remaining blocks. A block 200x200 ft was laid out, with four scrams spaced 50 ft apart for drawing and placed at a right angle to the strike. Finger raises were placed in a 25-ft interval grid pattern, with flat undercutting done by crosscuts at the undercut level 25 ft above

the scam. Remaining undercut pillars were blasted out on a diagonal retreat. No timbering was needed at the undercut level. The four scrams were timbered and lined with light rails, and there were six rounds of 10-in. cribbing in the fingers. No boundary weakening was done.

Ore to the mill was to be transported through the main adit level, over 2 miles long. It was anticipated that the caved ore would contain much coarse, blocky material and that underground crushing would therefore be necessary before the caved ore was hauled in mine cars designed primarily to handle the size of ore produced from square sets. A 36x48-in. jaw crusher was installed and set for about 5½-in. discharge size. The crusher is located to serve most of the block caving tonnage, collected by ore passes which eliminate any scraping except in the scrams. A conveyor belt 400 ft long carries crushed ore to the main ore pass, from which trains are loaded on the main mine level 500 ft below.

Scrams of Block 1 were equipped with 20-hp tuggers and 42-in. Eimco folding scrapers. A transfer drift, 10 ft lower than the scrams, provided for transfer to main ore passes by means of a 100-hp tugger and two 60-in. scrapers in tandem. This block was about one-third undercut (15,000 sq ft) when operations were closed by a strike late in August of 1949, and during the three months of the strike no work was carried on here. When work was resumed late in November it was impossible to draw a clean diagonal line at the pillar ends. It was apparent that caving had started in spite of the fact that the undercut area was relatively small. Undercutting of the remaining two-thirds of the block was completed and drawing begun. Despite very low metal prices (Pb 11¢ and Zn 9¾¢) drawing continued until March 1950 to provide necessary operational data. During the four months of drawing 125,000 tons, a record of drawing costs was obtained which confirmed previous estimates. Satisfactory answers were obtained to the important questions: Will the quartzite cave? What will be the size of caved material? It was proved that scam timbers were not subjected to weight. Had this block been operated longer, the rails used to line the scrams would have been replaced with the type of lining now used in the second and third blocks. A solution was still to be found, however, for planning scrams and drawing the orebody as the dip flattened to 65° and subsequently to 35°.

Block 2

The disadvantages of *standard flat undercut* when applied to a dipping orebody are very apparent, resulting in an unwelcome choice. If only the ore were undercut, a considerable wedge of ore up the footwall would be lost even if vertical draw-angle were possible. The alternative, a flat undercut in waste to reach under this footwall ore, would necessitate drawing large amounts of waste to reach ore.

It was decided to undercut and draw by means of scrams parallel to and under the ore footwall, an operation later discovered to be strikingly similar to that used by the Mufilira Copper Co. in Africa. This plan involved a stair-step pattern of scrams up the footwall, obviously risking the unknown effect of subsidence, since even a minor deviation, or *bulge*, in draw angle for drawing in a lower scam would destroy the scam above it.

The stringers of mineralization, and some shearing, were steep and somewhat parallel to the pro-

posed plane of the scrams. If these mineralized veins or the shearing were too weak, slippage along them would destroy the scrams. However, bedding and jointing of the quartzite, which is generally at right angles to the steep veins, proved to be of equal or greater weakness. At the end of 1952, with total production of 591,000 tons from Block 2, no subsidence or slipping has occurred in the plane of these footwall scrams.

Undercutting is accomplished by shrinkage on the ore footwall. After the scam is finished, finger raises are driven, connected, and belled out. Shrinkage begins and undercuts the ore footwall until the finger raises of the next successively higher scam are reached. Access to the shrink is from small temporary raises driven from the shrink bottom to hole out in the side of the scam above. Ultimately these temporary raises become part of the shrink as work progresses upward. This undercut method has been used successfully on the ore footwall where the dip is not less than 45°. Subsequent undercutting in future blocks with less than 45° dip will require some modification. However, the major fault, which is very weak, adjoins much broken ground, so that undercutting may not be required. The problem may well be to get finger raises in and belled before this fault starts to *run up dip*.

Each of these inclined blocks is served by an ore-pass and service raise. Block 2 was 200 ft long, measured horizontally along the footwall. Block 3 will be 300 ft long with the orepass and service raise in the center to serve 150 ft in each direction.

Scrams

Of particular interest is the evolution of scam linings. Rails have been tried in Block 1, but failed on impact of either the heavy scraper or large boulders *paying-off* from finger raises. Lighter scrapers of less capacity would result in fewer tons per manshift. Scrams were therefore modified to accommodate the Eimco folding scraper, which had proved very adaptable in mining Bunker Hill ore.

Scram sets are 12x14-in. timber, the posts 7 ft 8 in. long and the caps 8 ft long. Usually these sets are placed on 6-ft centers. End-grain blocks are placed between the sets and a shear wall is created which takes tremendous abuse. To protect the scam posts, blocks are about 2 in. longer than the post is thick, see Fig. 6. In addition to this, scam posts and caps are bolted together with threaded rods on the finger raise side of the scam. This practice helps to prevent finger posts being knocked out and holds end-grain blocks tight in sets immediately adjacent to the open set where the finger is located, see Fig. 7. To be successful, these end-grain blocks must be put in very tightly, with filling behind them if there are irregularities in the scam.

This type of scam has provided a lining which has cut repairs to a bare minimum. Once these end-grain blocks are installed properly no maintenance is required as the relatively coarse caved ore is drawn, see Fig. 8. No lining is used on the bottom of the scam. Scram posts are battered about 6 in. out at bottom. This has been sufficient to keep posts from *kicking in*. Grizzlies, at 2-ft centers, control size of material discharged to ore passes, Fig. 9.

The dimensions of the finger set are 6x5-ft and the maximum dimensions of irregular boulders handled by scraping are 3x3x2½-ft.

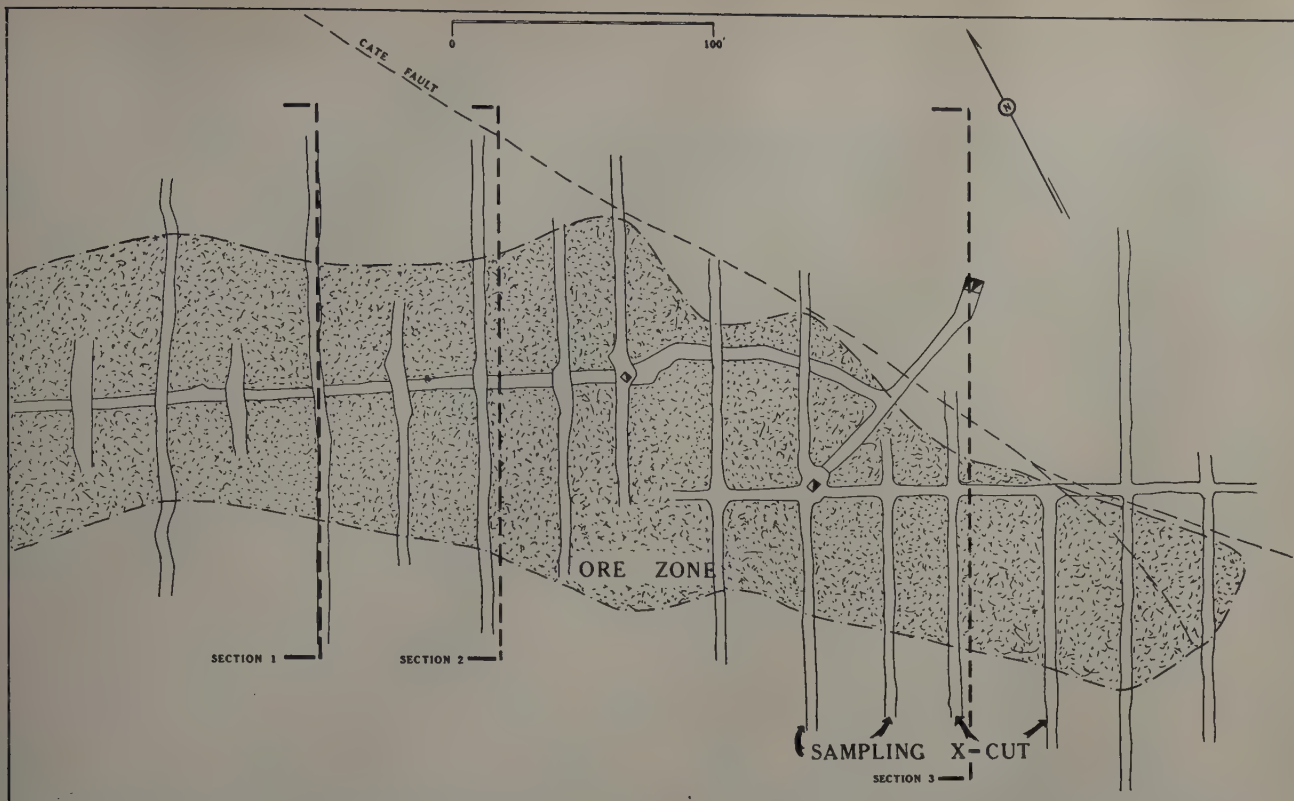


Fig. 1—Plan view of Bunker Hill orebody, 1000 ft long and 165 ft wide in horizontal section. The vertical extension is about 300 ft from an old level to the surface.

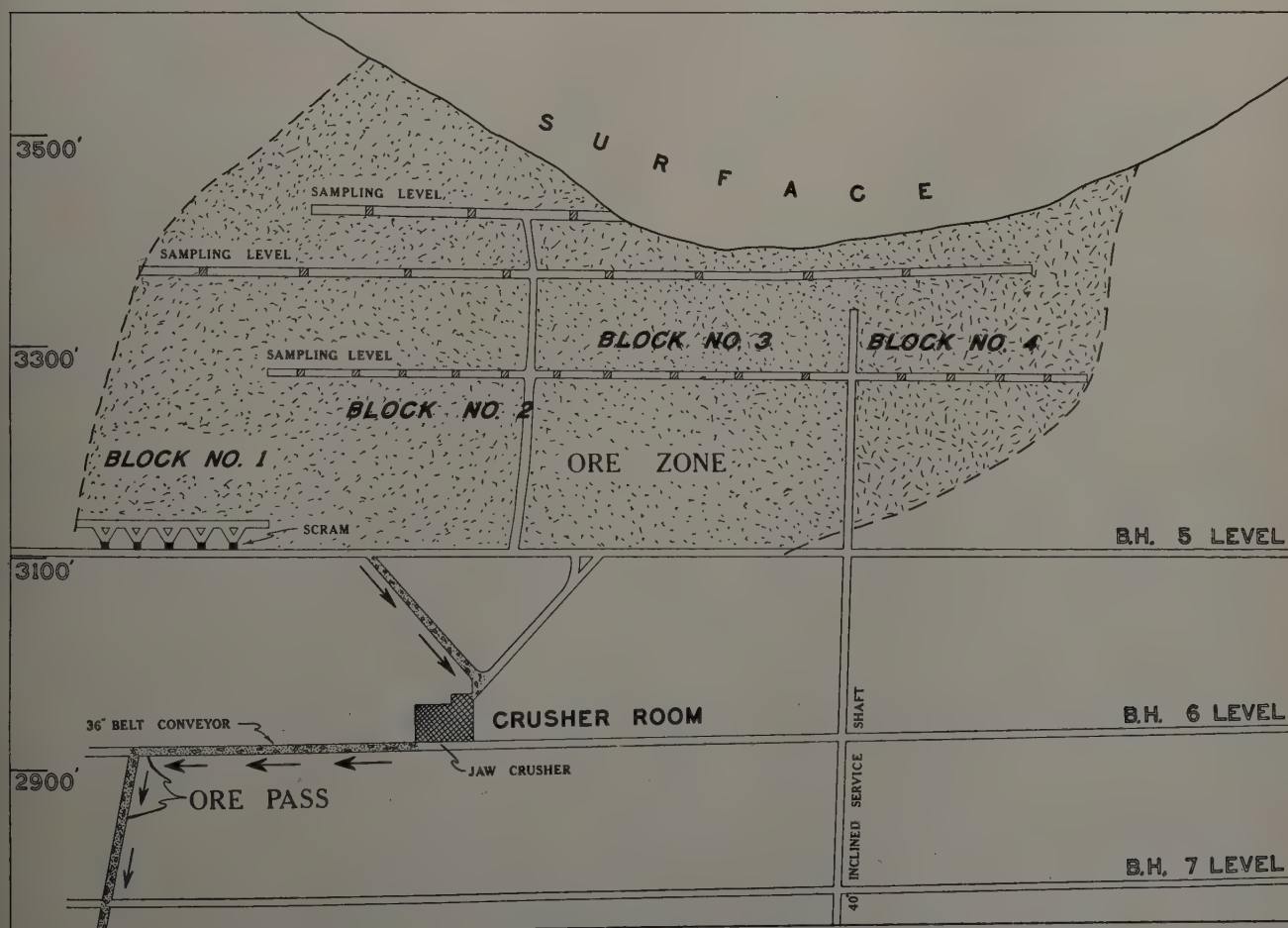


Fig. 2—The longitudinal projection for mining a section of the Bunker Hill orebody shown in Fig. 1.

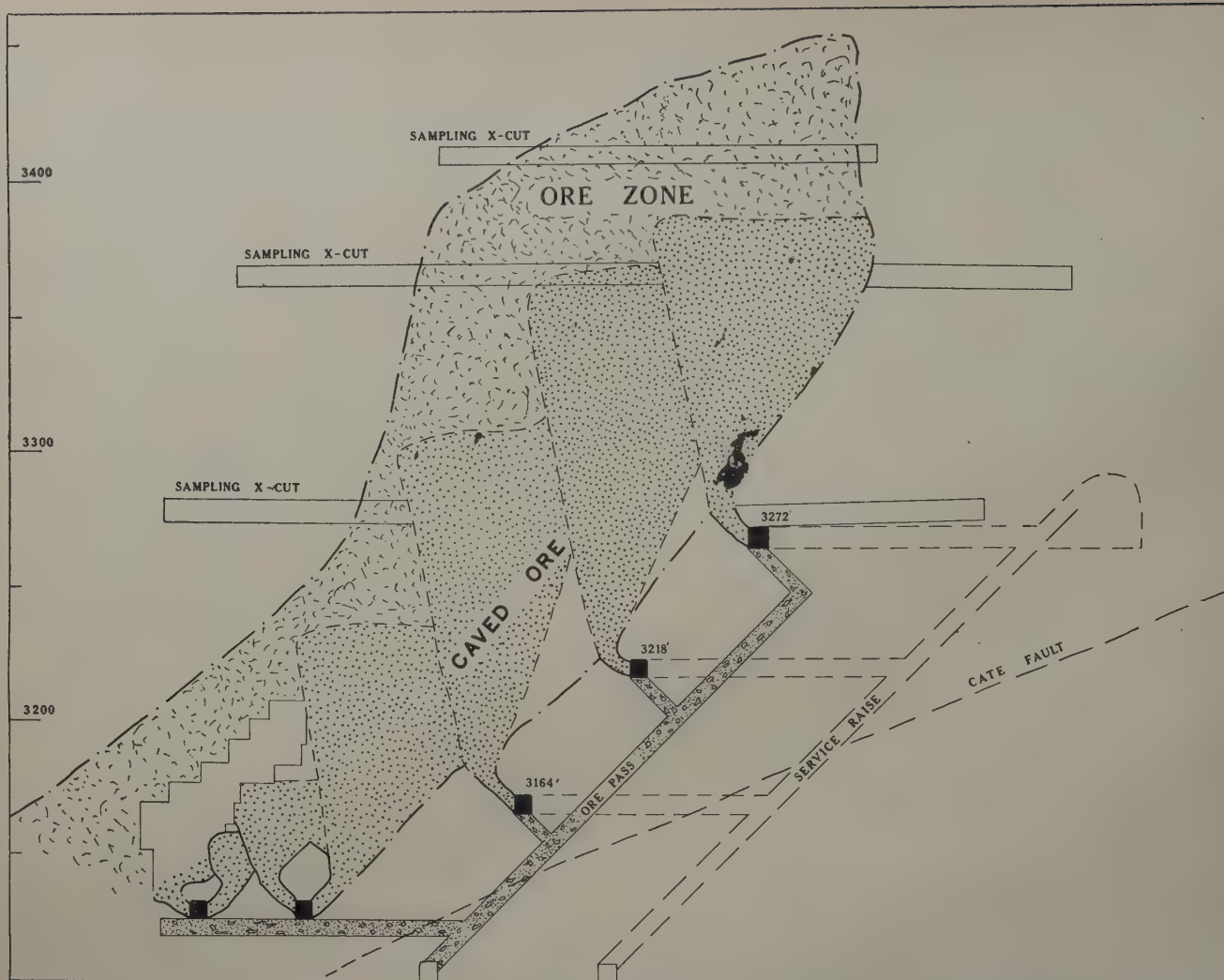


Fig. 3—Section 1 of the Bunker Hill orebody. The ore lies in the hanging wall of a major fault of the Bunker Hill mine. The wedge of waste which separates the ore from the fault pinches out near the center of the zone, where the ore dips 45°, lying nearly on the fault as shown in Fig. 4.

When coarseness of the caved ore and the relatively low daily production for a block-caving program are taken into account, this type of scam construction with 20-hp tuggers and the Eimco folding scraper has been the optimum combination from a standpoint of drawing costs and maintenance costs. There has been little wear on the end-grain blocks, and most of them can be used again after drawing is completed.

Concrete linings have been considered and may be used experimentally in future. The economic advantage of using concrete on the Bunker Hill job is doubtful, however, particularly if end-grain blocks can be used in two or three locations before being discarded.

Ore passes have also been lined with end-grain blocks where the condition of the ground demanded lining to control wear. These ore passes are generally lined with longer blocks, from 18 to 24 in., depending on the amount of rough, coarse ore that is to be handled.

Drawing

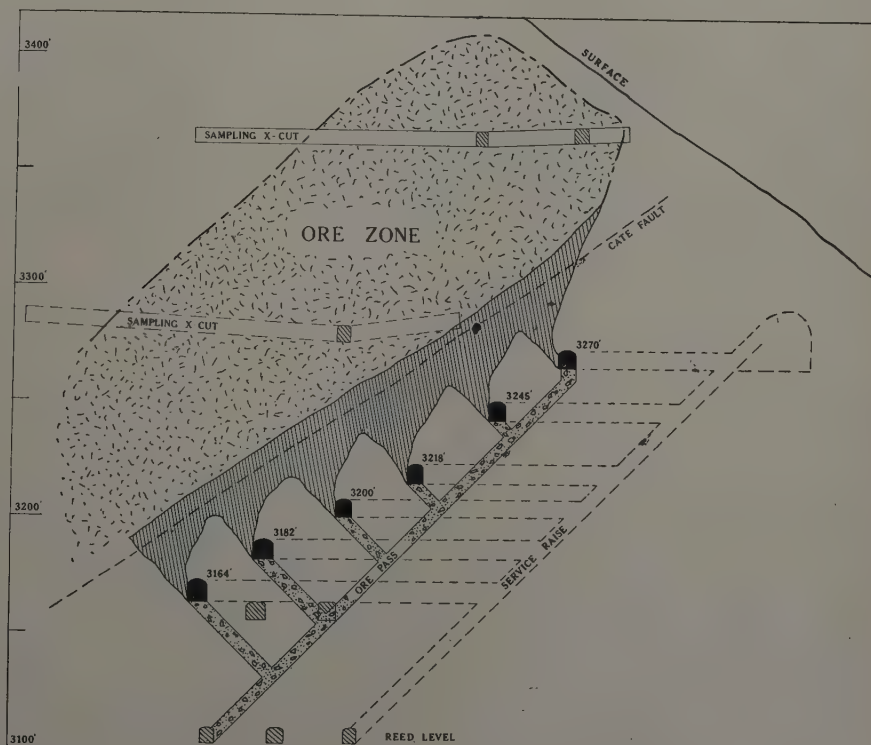
Draw control is used to maintain *ore-hangwall and waste* contact at about the same angle as it was originally. Rate of draw, measured horizontally, equals $\frac{1}{2}$ ft per day. If fingers do not *hang up* when drawn sufficiently, a few pieces of lagging will shut them off. Secondary blasting consumes 0.08 lb of

powder per ton of ore and blasting high hang-ups is done by means of dynamite tied to blasting slats. When hang-ups are more than 20 ft from the fingers, which happens rarely, they are blasted from the scam so that miners will have an opportunity to get away if the finger runs before anticipated. Ventilation to each scam comes from the service raises which are under pressure. Fresh air sweeps past the tugger and exhausts up small ventilation raises located at scam ends. The tugger man works in the fresh air stream and dust and smoke are driven away from him. High pressure water sprays are also used to minimize dust caused by scraping. The 8000 to 10,000 cu ft of air forced into each scam clear powder smoke of secondary blasting without much loss of production time. The single man who works in these producing scams draws an average 150 tons of ore per shift while doing all necessary blasting and making minor repairs.

At the time Block 2 was 40 pct drawn, undercutting of the adjacent end of Block 3 was begun and put under slow initial draw. The undercutting of Block 3 will continue at a rate calculated to bring in new fingers as those at the extreme end of Block 2 are finished. This results in a slow retreat across the whole ore zone and attempts to establish and maintain a broken ore-broken waste contact or plane which gradually moves along the strike of the ore at a constant angle. The angle of subsidence, or



Fig. 4 (above)—Section 2 of the orebody. The wedge of waste which separates the ore from the fault pinches out in this section, where the ore dips 45° , lying nearly on the fault. Fig. 5 (right)—Section 3 of the orebody.



Re-usable End-Grain Wood Blocks Protect Scram Walls

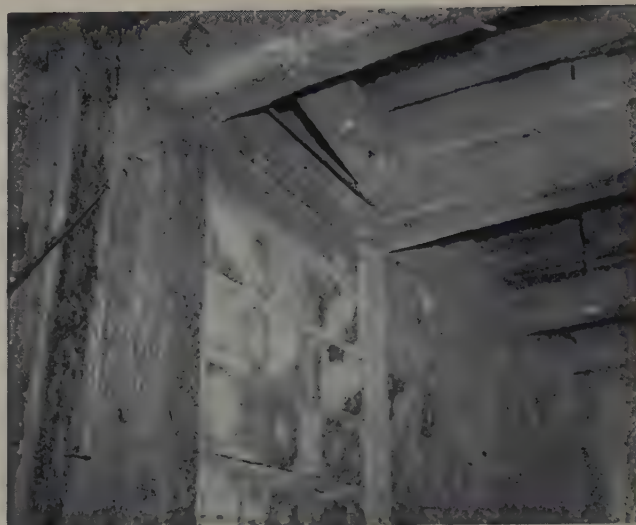
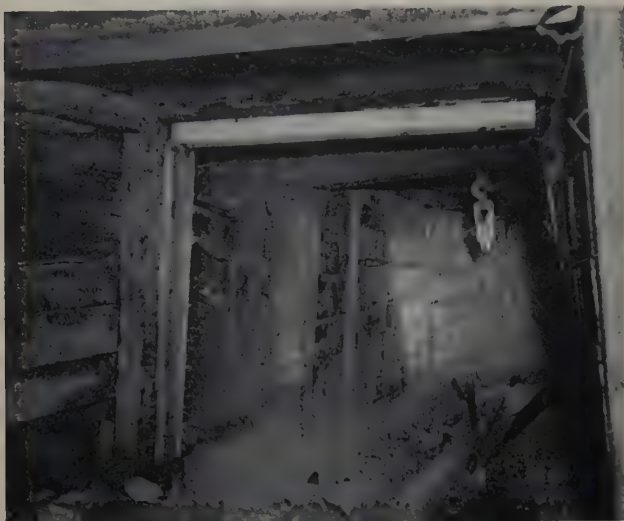


Fig. 6 (left)—To protect the scram posts, blocks are set about 2 in. longer than the posts are thick. Scram posts and caps are bolted together with threaded rods on the finger raise side of the scram. Fig. 7 (right)—End grain blocks must be put in very tightly, with filling behind them if there are irregularities in the scram.

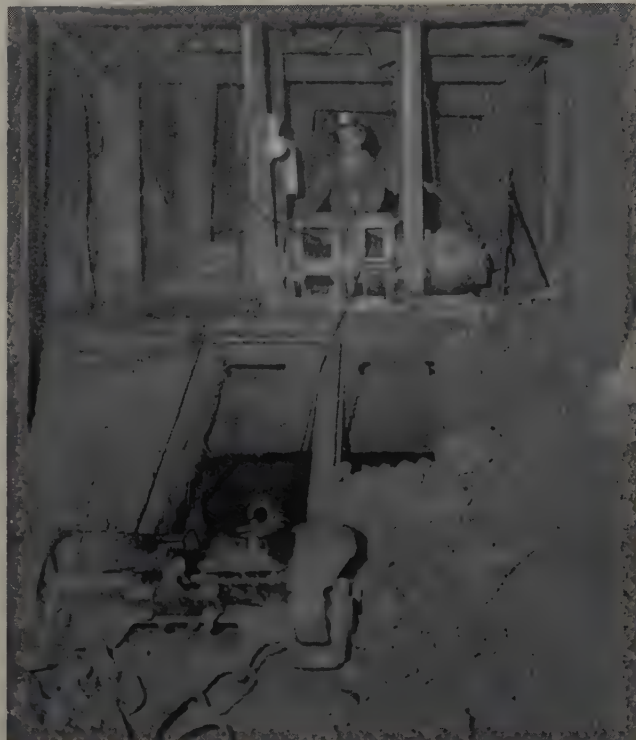


Fig. 8 (left)—As shown above, caved ore is comparatively coarse. Fig. 9 (right)—Grizzlies at 2-ft centers control the size of the material discharged to the ore passes.

angle of draw, has been measured as 82° to 83° inclined into the block.

No calculations of drawing efficiency are available at this time, since no blocks are completely drawn. Also, such calculations are dependent on metal prices. For instance, Block 1 at high metal prices still contains ore, and since there is no weight on the scrams, its openings are still in good condition ready for additional drawing if market conditions warrant it. If the scrams had taken weight, as normally occurs in many block-caving operations, it would have been impossible to maintain economical operations with the violent fluctuations in lead and zinc prices in the last four years.

For the first 10 months of 1952 the total cost per ton was \$3.97 for all mining, transportation, and

milling. This figure covers all chargeable expense, excluding only general plant depreciation and corporate income taxes. Although today's very low lead and zinc prices forced curtailment of work February 7, at reasonable market prices these operations are highly successful. There is little doubt that they have played some part in the evolution of a method of block-caving peculiarly adaptable to dipping orebodies, and it is almost a certainty that this first orebody is a fair sample of the type of occurrence to be expected in other areas at Bunker Hill mine.

In this presentation it has been possible to discuss only the general facts concerning block caving at Bunker Hill mine. Inquiries are invited from those interested in additional data.

Alkali Reactivity of Natural Aggregates In Western United States

by William Y. Holland and Roger H. Cook

In view of the increasingly widespread deterioration of concrete structures as the result of the interaction of the alkalis sodium and potassium released by hydration of portland cement and susceptible rocks and minerals in aggregates, it is believed that a paper summarizing the geographic distribution of these aggregates will be of interest to producers and users of concrete, concrete products, and concrete-making materials.

THIS paper reviews the problem of alkali-aggregate reaction in concrete and describes the geologic and geographic occurrence and distribution of alkali-reactive sand and gravel in western United States. It includes no discussion of crushed stone or synthetic aggregates.

Most deposits of sand and gravel are accumulations of particles of rocks and minerals from a variety of sources, and it is not unusual for at least one or two varieties of the rocks to contain some form of reactive material. Examination by petrographic methods of many sands and gravels, as well as manufactured aggregates, has shown that a comparatively high proportion of the deposits does contain, in greater or lesser degree, rocks and minerals known to be deleteriously reactive with the alkalis of cement. Fortunately the amount of reactive materials is commonly less than that necessary to cause deleterious effects in concrete.

As investigation of unsound concrete structures progresses, it becomes evident that the alkali-aggregate reaction is even more widespread than supposed, Figs. 1 and 2. Even though some parts of the country appear at present to be immune, further investigations will probably show that the effects of alkali-aggregate reaction can be seen in many structures in these areas, although only on a small scale in most of them. Many concrete structures will, of course, have lived their useful life before disintegration from this cause is serious, and in others the alkali-aggregate reaction may never become significant even though the microscopic evidence of reaction is present.

The alkali-aggregate reaction first was reported to be a cause of deterioration of concrete in 1940 when Stanton¹ described expansion of concrete pavements in California. Similar expansion and deterioration of concrete was recognized during succeeding years in concrete structures located in many parts of the country, but particularly in the

western states. A number of concrete laboratories² became interested in the problem. It was soon determined that only certain combinations of aggregate and cement caused the alkali-aggregate reaction to take place, and moreover that the reaction progresses only in the presence of water. Further research proved that cements containing more than 0.60 pct total alkalis (pct $\text{Na}_2\text{O} + 0.658 \times \text{pct K}_2\text{O}$), when used with aggregates containing appreciable amounts of reactive ingredients, caused the reaction to take place, usually with subsequent deterioration of the concrete.

During the last few years this limitation has been adhered to in both government and private construction as the maximum allowable alkali content of cement to be used with aggregates of known alkali reactivity. Because of this limitation, it appears that deleterious reaction either has been reduced or eliminated in many recently built structures in which it probably would otherwise have occurred.

Recent tests have shown that the degree of expansion obtained with any particular cement-aggregate combination depends not only on the alkali content of the cement but also upon the relation of this alkali content to the amount and degree of reactivity of reactive constituents in the aggregate.³ In laboratory mortar bars, opal and cements with alkali content of as low as 0.2 pct (as equivalent of Na_2O) have produced deleterious expansion as the result of alkali-aggregate reaction. These experiments demonstrate that aggregates containing even 0.1 pct of opal are deleteriously reactive.

It was soon determined that alkali-silica gels were formed by the interaction of the alkalis of the cement and the reactive aggregate, Figs. 3 and 4. Osmotic or swelling pressures produced by the continued hydration of these gels cause expansion of the concrete with resulting cracking, warping, and dislocation.

Evidence of the alkali-aggregate reaction can be seen by a petrographic study of the deteriorated concrete. Among the first structures studied by this method was Parker Dam on the Colorado River, California-Arizona. In the concrete from this dam pebbles of rhyolite, andesite, siliceous limestone, and chalcedonic chert were found to be reactive.

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Discussion on this paper, TP 3663H, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, May 11, 1953. Revised Aug. 19, 1953. Los Angeles Meeting, February 1953.



Fig. 1—McCook air base, Nebraska. Expansion of concrete has resulted in extreme buckling at expansion joints.

Reaction appeared to be progressive, characteristically developing a rim extending from the exterior of the pebbles toward the interior. In many examples the whole pebble appeared to have reacted, in others only a rim of varying thickness. On close examination the gels formed by the interaction of alkalis of the cement and the volcanic glass, opal, and chalcedony of these rocks could be seen in the immediately surrounding mortar, within the aggregate, and in microfractures radiating from the reacted pebbles, Figs. 5 and 6. Similar microscopical evidence of alkali-aggregate reaction has been found in association with other known reactive materials.

Ameliorative Measures

Considerable progress has been made toward the development of materials and methods that retard or even eliminate the alkali-aggregate reaction. One of the first ameliorative measures taken was to limit the amount of alkalis in cements that are to be used with aggregates of known reactivity. It has been found that the use of various pozzolans such as the opaline shales and cherts of the Monterey formation of southern California will retard the expansion. Air-entraining agents in certain proportions also reduce the expansion resulting from this action. With air-entraining agents, the chemical reaction may progress, but the products of the reaction (the gels) probably find room for their volume increase in the air voids. The addition of small amounts of certain materials such as lithium carbonate and some proteins to high alkali cement have an inhibiting effect on the expansion reaction.⁴

Alkali-Reactive Materials

Identification of reactive materials in an aggregate can be made petrographically. Such examinations are now routine in many concrete laboratories. A competent petrographer can make the examination in a comparatively short time.^{5,6} The usual procedure is to examine and identify the pebbles from a sample of each fraction of gravel and sand under a binocular microscope. If more detailed observation is necessary for the identification, petrographic thin sections or mounts of crushed particles are examined under a petrographic microscope. Each fraction is separated into its constituent lithologic groups on the basis of this examination and the percentages are calculated.

If reactive ingredients are disclosed, further study is made to establish the amount and probable degree of reactivity of these constituents.

Aggregates now known to be deleteriously reactive contain one or more of the following, as shown also in Table I: Acidic to intermediate volcanic rock (e.g., glassy to cryptocrystalline rhyolites, latites, dacites, andesites and their tuffs, and glassy rocks such as obsidian, perlite, and pumice); the silica minerals opal, chalcedony, tridymite, and cristobalite; the zeolite heulandite, and probably ptilolite and certain phyllites. Any rock containing a significant proportion of reactive substances will be deleteriously reactive; thus, although pure limestone and dolomites are not reactive, opaline or chalcedonic limestone and dolomites are deleteriously reactive. Similarly, any normally innocuous rock can become deleteriously reactive if coated or impregnated with opal, chalcedony, or other reactive substances.

Based on investigation of aggregates of known deleteriousness, estimates can be made of proportions of various constituents necessary to produce distress in concrete. Aggregates containing more than 0.25 pct by weight of opal, more than 5 pct of chalcedony, or more than 3 pct of glassy or cryptocrystalline acidic to intermediate volcanic rocks are regarded by the Bureau of Reclamation as harmfully reactive. These limits are probably conservative; to the extent that reactive ingredients are admixed or enclosed within innocuous substances the limits can be increased. For example, opal may occur in isolated masses within innocuous material and will be less susceptible to alkali attack.

The acidic to intermediate volcanic rocks owe their reactivity primarily to the natural volcanic glass, Fig. 7, of which they are wholly or in part composed. Volcanic rock may range from a wholly glassy material, such as perlite and obsidian, to a fine-grained porphyritic variety, such as andesite porphyry. The extent to which a volcanic rock will contribute to alkali reactivity depends on the amount of natural glass or cryptocrystalline material in its ground mass. Particles of a reactive volcanic rock type may be only slightly deleterious or even innocuous if the original glass is completely recrystallized as a result of geologic processes. The natural glasses are an amorphous substance usually containing from 70 to 75 pct silica in the acidic or rhyolitic varieties and from 50 to 65 pct in the intermediate or andesitic varieties. Basaltic glasses containing 45 to 50 pct silica are not deleteriously reactive with cement alkalis. Microscopical observations over extended periods of samples of glassy tuffs mixed as a slurry in alkaline solutions, such as occur with hydrating high-alkali cements, have shown that the glass slowly reacts with, or dissolves in, the alkaline solutions and forms alkalic-silica gels.

Opal, opaline cherts and shales, and other rocks containing opal owe their reactivity to the amorphous, hydrous silica (opal) of which they are composed wholly or in part, Fig. 8. It has been shown that amorphous hydrous silica combines readily with the alkaline solutions of cement to form gels which are clear at first but ultimately turn white.

Rock containing certain zeolite minerals can contribute to alkali-aggregate reaction indirectly through cation-exchange reactions of the zeolites.

Sodium and potassium which are thus released augment the supply of alkalis from the cement. Tests have been made on analcite-bearing phonolite porphyries from Missouri Buttes and Barlow Canyon, Wyo., and on a basalt from Table Mountain, near Golden, Colo., which contain several zeolite minerals, including analcite, chabazite, and natrolite. These tests indicate that alkalis are released by cation-exchange reactions between the zeolites and calcic solutions produced by the hydrating portland cement. Examination of mortar bars fabricated with these materials indicated that the rocks themselves do not cause deleterious expansion if used as the entire aggregate, even with high-alkali cement. On the other hand, when ground and used as an addition or partial replacement of low-alkali cement in mortar containing alkali-reactive aggregate, the rocks release sufficient sodium and potassium to cause significant expansion of mortar.⁹

Recent studies¹⁰ with the electron microscope have shown that chalcedony probably consists of fine-grained quartz and a variable quantity of free water; no opal could be identified. If chalcedony is composed essentially of fine-grained quartz, its reactivity must result from the extremely high surface area available for attack by alkalis. It is known that pulverized quartz releases considerable silica to alkaline solutions, whereas massive quartz releases comparatively little. Moreover, recent investigations¹¹ show that particles of quartz dust possess a highly soluble surface layer and that this layer can be removed easily by various solvents with the formation of a flocculent colloidal silica.



Fig. 2—Example of alkali-aggregate reaction, Coolidge Dam, Arizona. Crack in pier on left end of upstream parapet wall. Maximum opening of crack, 1½ in.

Much has yet to be learned about the reactivity of the other minerals mentioned in Table I. However, they undoubtedly perform deleteriously in a

Table I. Alkali-Reactive Rock and Minerals

REACTIVE MINERALS	COMPOSITION
Opal	SiO ₂ .nH ₂ O
Chalcedony	SiO ₂
Tridymite ⁷	SiO ₂
Cristobalite ⁸	SiO ₂
Heulandite	H ₄ CaAl ₂ (SiO ₃) ₆ .3H ₂ O
REACTIVE ROCKS	
Siliceous Rocks	
Opaline cherts	Opal
Chalcedonic cherts	Chalcedony
Siliceous limestones	Chalcedony and/or opal
Volcanic Rocks	
Obsidian, perlite, pumice	Acidic to intermediate volcanic glass
Rhyolites and rhyolite tuffs	
Latites and latite tuffs	
Dacites and dacite tuffs	
Andesites and andesite tuffs	
Miscellaneous Rocks	
Any rocks containing veinlets, inclusions, or detrital grains of the reactive substances listed above.	

manner similar to the natural glasses and amorphous hydrous silica, that is, as a consequence of attack by alkaline solutions, they produce a gelatinous substance that increases in volume by imbibition of water.

Alkali-Reactive Aggregate Deposits

Although alkali-aggregate reaction has been identified in concrete structures in the eastern and southeastern parts of the United States, alkali-reactive aggregate is of particular concern to users and producers of concrete and concrete products in the western states because of the widespread occurrence of potentially reactive rocks and minerals. The map, Fig. 9, shows the geographic distribution of some 245 deposits which were investigated in the Bureau of Reclamation laboratories and found to contain deleterious proportions of reactive rock types. Most of these deposits are located in the drainage basins of the Missouri, Platte, Rio Grande, Arkansas, Columbia, Snake, Colorado, Sacramento, and San Joaquin Rivers.

Only deposits of reactive aggregates are shown on the map. However, this is not intended to imply that all aggregate deposits in these areas are reactive. On the contrary, of the many additional aggregate deposits investigated in these river basins, some were composed wholly of innocuous rock types; others were found to contain reactive rock types in amounts less than those considered necessary to produce a harmful degree of reaction in concrete.

The lithologic composition of a specific aggregate deposit is the result of a complex geologic history. Deposits of natural gravel and sand have their origin, first, in the weathering and erosion of rock formations. Secondly, the particles of ledge rock produced by erosion are transported from their point of origin to their present location. The major transporting agencies, in the case of natural aggregate suitable for use in concrete, are streams and glacial ice.

The lithologic composition of gravel and sand deposits is controlled by the rock types available at the sources. If formations composed of potentially reactive rock, such as a glassy rhyolite lava flow or a siliceous limestone, occur within the drainage basin of a river, it is inevitable that at least

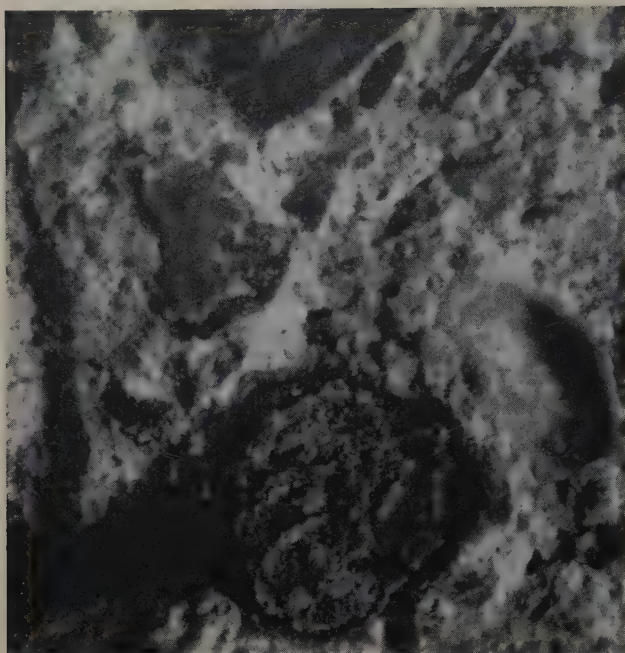


Fig. 3 (left)—Photomicrograph shows partially decomposed pebble, gel-lined void, and accompanying gel in concrete. From a highway in California. Original magnification X25. Fig. 4 (right)—Photomicrograph of alkali-silica gel bead extruded from laboratory concrete. Original magnification X6.

some of the reactive rock will be eroded and find its way into aggregate sources along the river. The proportion of reactive rock types in an aggregate will depend, at least in part, on the relative abundance of reactive rock formations in contrast to nonreactive rock formations, as well as the relative ease with which each type of rock can be eroded, the kind and severity of the transporting processes involved, and the distance from the source.

Tributaries to major streams contribute to local variations in kind and amount of rock types occurring along the main streams and may thus influence the reactivity of the aggregates. If, for example, a tributary flows through an area composed predominantly of potentially reactive andesitic rock, the ag-

gregate along the tributary will, of course, be predominantly andesitic and reactive. Hence the concentration of reactive andesites in sand and gravel carried by the main stream will be increased below the confluence of the streams. The seasonal variations in the amount of sediment transported by streams may repeatedly alter the lithology of deposits.

Terrace deposits frequently present special problems. Terrace deposits of aggregates deposited by ancient streams and glaciers are often eroded and redeposited as a result of changes in course of present-day streams; deposits so formed will differ lithologically from deposits along the same stream formed from bedrock material. Sand and gravel in

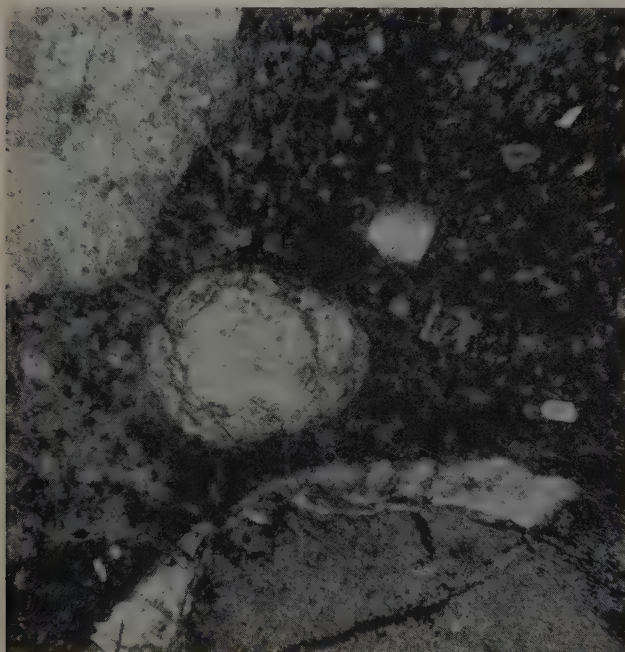


Fig. 5 (left)—Photomicrograph of gel-lined void in thin section of concrete from Stewart Mountain Dam, Arizona. Ordinary light. Original magnification X120. Fig. 6 (right)—Photomicrograph of gel-filled crack in thin section of concrete from Stewart Mountain Dam. Ordinary light. Original magnification X120.

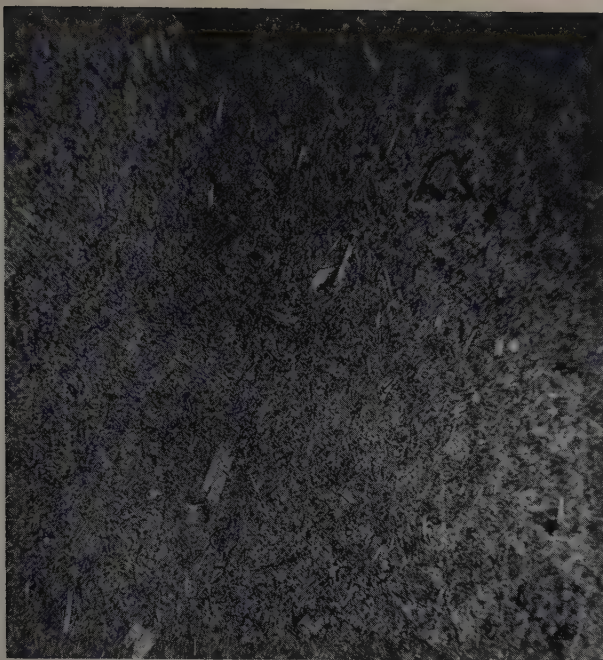


Fig. 7—Photomicrographs of a thin section of potentially alkali-reactive glassy rhyolite. Ordinary light on left and crossed nicols right. The dark phase appearing in the right photomicrograph is the glass constituent. Original magnification X120.

terrace deposits may be impregnated or coated with opaline or chalcedonic material brought in by percolating water; the aggregates from these deposits thus would be alkali-reactive, even though the area is free from formations composed of alkali-reactive rocks. Some terrace deposits may contain volcanic or other reactive rock types derived from ancient and now nonexistent sources and hence will differ from sand and gravel deposited along the existing channel.

Fundamentally, for alkali-reactive rocks to be present in an aggregate deposit, formations capable of supplying the reactive rocks and minerals must be present in the source areas for that deposit. In the 17 western states such potential source areas of reactive materials are abundant. Geographic distri-

bution of volcanic lava flows in the western states is shown on the map, Fig. 10. The shaded areas indicate volcanic rock flows in which rhyolites, trachytes, latites, dacites, andesites, and their related tuffs are reported to occur. It is not intended to imply that all the rocks in the areas shown have been proved to be capable of producing harmful reactions with cement alkalies. The potential deleteriousness of the rock in each of these areas would have to be ascertained individually by field and laboratory investigations. However, from the petrographic nature of these rocks the probability is high that glassy or cryptocrystalline phases capable of reactivity are present.

During the Miocene period of geologic time most of the Columbia River plateaus, which include the

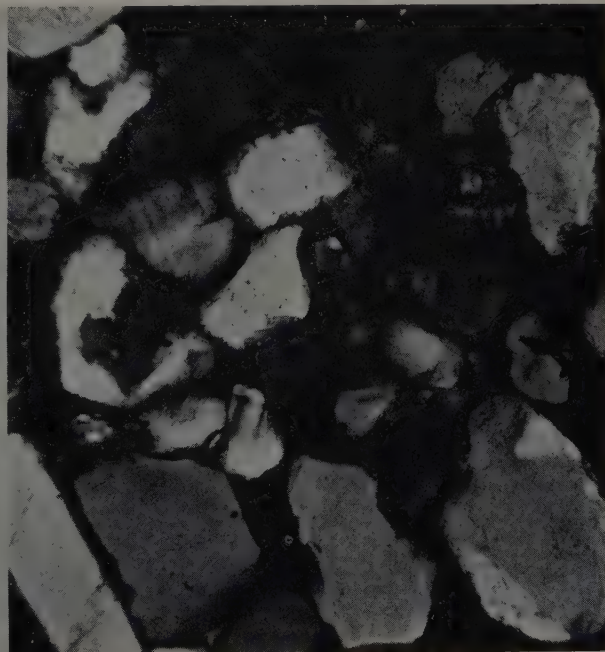
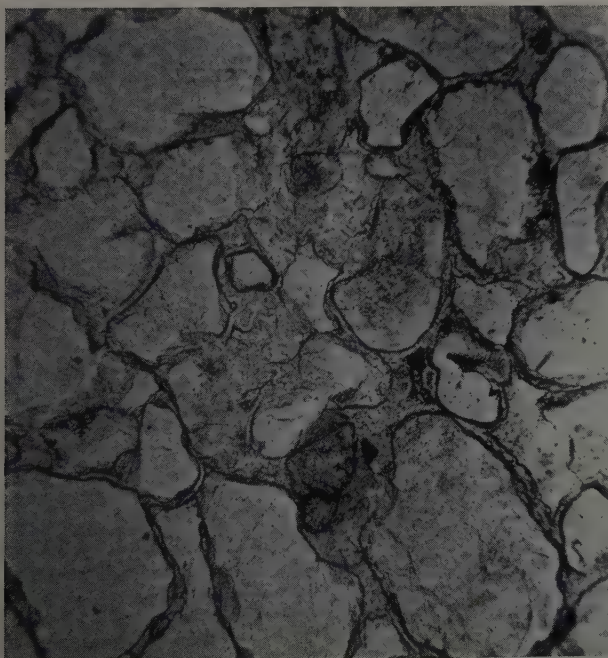


Fig. 8—Photomicrographs of a thin section of potentially alkali-reactive opaline sandstone. Ordinary light on left and crossed nicols right. The dark phase between the sand grains in the photomicrograph on the right is the opaline material. Original magnification X120.

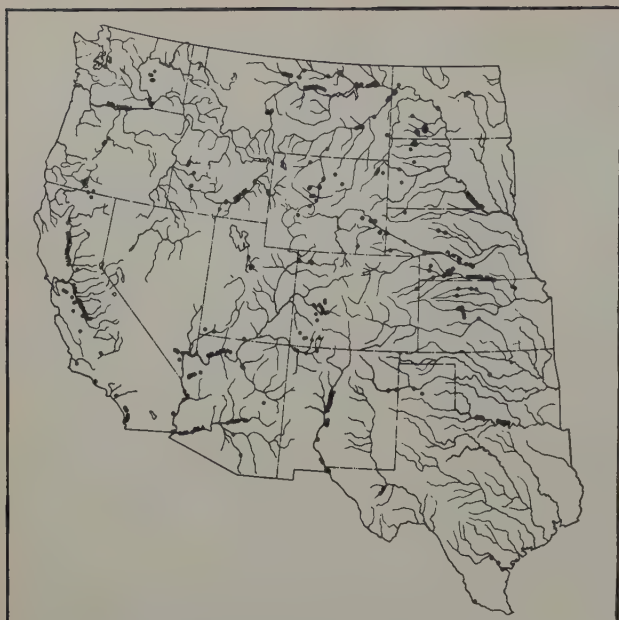


Fig. 9—Alkali-reactive aggregate deposits in western United States.¹⁰

central and eastern portions of Washington and Oregon and the western edge and southern portion of Idaho, were covered by thick flows of basalt. Consequently, the aggregate deposits in this province are composed predominantly of basalts and with rare exceptions are not reactive, unless as in some of the deposits in the Columbia Basin area of central Washington, the basalt particles are heavily coated with opal. Much smaller flows of andesite and rhyolite have occurred during earlier and later geologic periods in several localities within this province. These flows contribute alkali-reactive materials to local deposits. For example, the Deschutes River in central Oregon has cut through the Columbia River basalt, exposing the Clarno formation of lower Oligocene and upper Eocene age. This formation, locally as much as 2000 ft thick, is composed of aphanitic to glassy andesites, glassy rhyolites, and perlites, with interbedded layers of tuffs, ash, and breccia. Results of alkali-aggregate reaction have been recognized in concrete of a highway bridge, Fig. 11, across the Deschutes River near Redmond, Ore.

Andesitic to rhyolitic volcanic rock flows are scattered throughout the Pacific Coast province, which includes the western portions of Washington and Oregon and most of California. In addition to the volcanic rock, sedimentary formations, such as the Franciscan and Monterey formations in California and the Oligocene and Miocene formations along the coasts of Washington and Oregon, have contributed opaline and chalcedonic rock to the various aggregate deposits.

Evidence of alkali-aggregate reaction has been recognized in concrete, Fig. 11, at the Hetch Hetchy power plant, in a railroad bridge near Friant Dam, in highway structures near Fresno in the San Joaquin Valley, in a flume structure in the southern Sierra Nevada, in buildings and structures in and near Los Angeles, in highway structures and sea walls at and near Santa Barbara and Ventura, and in highway structures near Bradley and Paso Robles in the Salinas Valley.

Tertiary volcanic activity was widespread in the Great Basin geologic province, which includes the

eastern edge and southeastern portions of California, all of Nevada, the western edge of Utah, the western edge and southern half of Arizona, and the southwest corner of New Mexico. Flows of acidic to intermediate volcanic rock are abundant in this area, some of which contributed reactive rhyolites, dacites, andesites, and trachytes to the aggregate deposits along the lower Colorado River. Particles of chalcedonic cherts and siliceous limestones contributed by various sedimentary formations in the area are found in the Colorado River aggregate deposits. Aggregates from terrace deposits along the lower Colorado River were used in concrete at Hoover and Davis Dams and Parker Powerhouse, Fig. 11, constructed by the Bureau of Reclamation, and at the Gene wash and intake pumping plants of the Metropolitan Water District of southern California. Aggregates used in the construction of Parker and Copper Basin dams were obtained from a terrace deposit along the Bill Williams River, near its junction with the Colorado River. Expansion in excess of 0.1 pct linearly has been measured in concrete at Parker Dam. Gene Wash and Copper Basin dams are affected by alkali-aggregate reaction. The aggregates used at Davis Dam proved to be deleterious in laboratory tests of concrete and mortar containing high-alkali cement. Petrographic analysis of aggregates used at Davis Dam identify the reactive constituents as rhyolites, dacites, andesites, chalcedonic cherts, and siliceous limestones. These rock types constitute about 45 pct of the analyzed gravels. Pozzolan material was used in construction of Davis Dam to relieve this condition.

The Rocky Mountain and Colorado Plateau province, which includes the eastern part of Idaho, western Montana, Wyoming, and Colorado, northwestern New Mexico, northern Arizona, and eastern Utah, contains numerous flows of acidic and intermediate volcanic rock. Such flows along the Continental Divide in northwestern Wyoming, southwestern Montana, and eastern Idaho, have contributed potentially reactive rock types to the drainage basins of both the Missouri and Snake Rivers. An aggregate considered for use at Blue Mesa Dam in

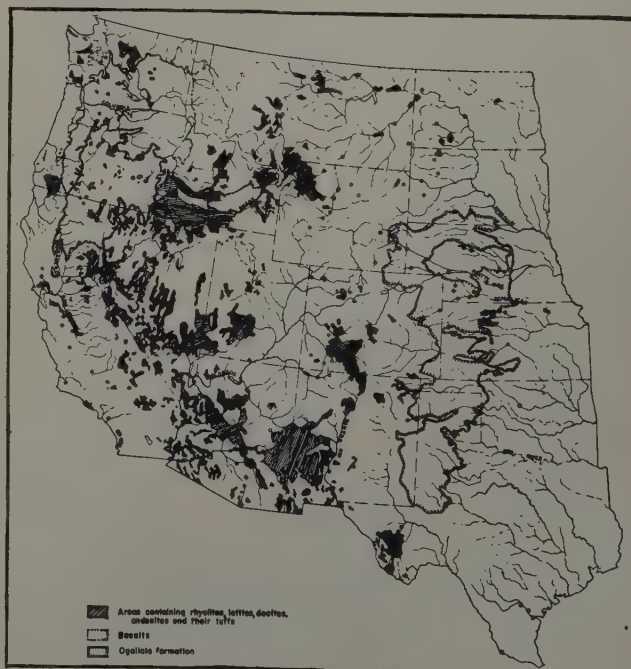


Fig. 10—Distribution of alkali-reactive aggregate deposits and lava flows in western United States.¹²

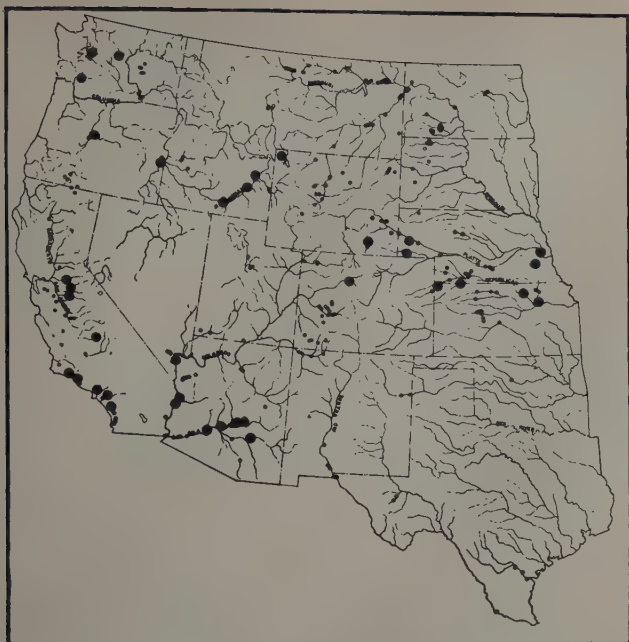


Fig. 11—Alkali-reactive aggregate deposits (small dots) and location of structures (large dots) that have been affected by alkali-aggregate reaction in western United States.¹²

western Colorado contains from 27 to 50 pct of rhyolites, andesites, tuffs, and chalcedonic sandstones. Mortar bars made from this aggregate and high-alkali cement show expansions of up to 0.473 pct in 12 months. Sedimentary formations containing reactive material are common in the area, such as the Bozeman formation in southwestern Montana, which contains rhyolites and opaline and chalcedonic rocks. Opaline and chalcedonic sandstone and limestones are found in southwestern Wyoming. The Marble Canyon limestones in northern Arizona contains chalcedonic zones.

Volcanic activity is virtually absent in the Great Plains province which includes eastern Montana, Wyoming, Colorado, and New Mexico, western Oklahoma, and Texas. However, the area is covered with large sedimentary formations, such as the Ogallala, which contain potentially reactive rocks and minerals. A portion of the Ogallala formation is composed of an opaline sandstone which contains up to 40 pct of opal as cementing material. Since many of the rivers in this area have their headwaters in the Rocky Mountain region, some volcanic rocks have been carried into this area. Volcanic glass particles present in some formations in this area probably are the result of explosive volcanic activity in the Rocky Mountains, particularly from the Yellowstone Park area. Evidence of alkali-aggregate

reactions has been found in concrete in pavements and curbs at Kimball, Neb.; air-strip pavements at McCook, Neb.; and in highway and other structures at Lincoln and Gretna, Neb., and Saint Francis, Norton, Clay Center, and Concordia, Kan., see Fig. 11. The aggregates used in these structures were obtained from the Platte and Republican Rivers or their tributaries.

New evidence of alkali-aggregate reaction continues to be found in old concrete structures. Only recently two large 40-year-old bridge slabs in western Missouri were examined and found to contain effects of the reaction. In this case the alkali-aggregate reaction was augmented by freezing and thawing. This concrete contained chert aggregate which, as previously mentioned, has been found to be reactive. Studies of affected structures and continued research on ameliorative measures to control the reaction are in progress and will help eliminate this troublesome factor in concrete engineering.

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- ¹² U. S. Geological Survey: Geologic Map of United States.

Corrections

In the June 1953 issue: TP 3554A. Testing of Roof-Bolting Systems Installed in Concrete Beams. By Rudolph G. Wuerker. On p. 608, col. 1, Abram's formula should read:

$$S_{\text{compr.}} = \frac{14,000}{7 \frac{w}{c}}$$

In the July 1953 issue: TP 3544F. The Blending of Western Coals for Production of Metallurgical Coke. By John D. Price. P. 718, Fig. 2, caption should read: Effect of blending coals of similar rank. Colorado high-volatile coals. Fig. 3, caption, should read: Effect of blending high and low-volatile coking coals. Colorado high-volatile coal, above, and Utah high-volatile coal, below. P. 719, Fig. 4, caption, should read: Effect of use of char. Colorado high-volatile coal, above, and Utah high-volatile coals, below. Fig. 5, caption, should read: Effect of use of coal tar pitch. Colorado high-volatile coal, above, and Utah high-volatile coal, below. P. 720, Fig. 6, caption, should read: Effect of use of petroleum coke. Colorado coal plus petroleum coke.

Orientation of Cube Diamonds in Drill Bits

by Eugene P. Pfeider

Research indicates that the cube diamond is a superior stone for drill bits. It can be oriented to combine a long cutting edge with structural strength and high resistance to wear. Test bits set with preferred orientation give double the bit life at astoundingly low diamond loss in comparison to oriented octahedral diamonds.

A PROGRAM of research on the orientation of diamonds in drill bits has been under way for several years at both the United States Bureau of Mines Experimental Mine at Mount Weather, Va., and the School of Mines and Metallurgy of the University of Minnesota. Reports on past work done both in the laboratory and the field are available in the technical press.^{1, 3, 4} The results of such drilling, done under comparable conditions, show that bits having diamonds oriented in a direction that combines structural strength and greatest hardness not only can advance somewhat faster than random set bits but can do this with a unit diamond loss of 50 pct or less. Several bit manufacturers have initiated the training of diamond setters in the principles of orientation on the basis of the above-mentioned research results.

The basic information on the crystallography and hardness variation of the diamond was presented in

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Discussion on this paper, TP 3660A, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, June 4, 1953.

an earlier paper,¹ which pointed out that the diamond is several hundred times harder along certain directions and certain faces than on others. The same article discussed the drilling results obtained with a group of nine bits having octahedron diamonds set with *on-point*, positive (*gouge*) rake, and negative (*drag*) rake orientations. The testing of oriented cube diamonds is an extension of this research program, employing the same procedures and equipment (Hy Mac hydraulic drill unit, EXT diamond core bits, St. Cloud Pink granite) as previously described.

This current work covers the test experience of seven bits set with cube diamonds running approximately 10 stones per carat. The individual stones were near perfect cubic form; hence it was simple to orient them in any desired direction. The cube is a comparatively rare crystal form of the diamond, although considerable numbers are produced from the Congo. It is understood that most of the cube diamonds are retained in Africa, where they are either used locally for diamond bit manufacture or ground up to make powder. They have been considered to be an inferior stone for drilling purposes.

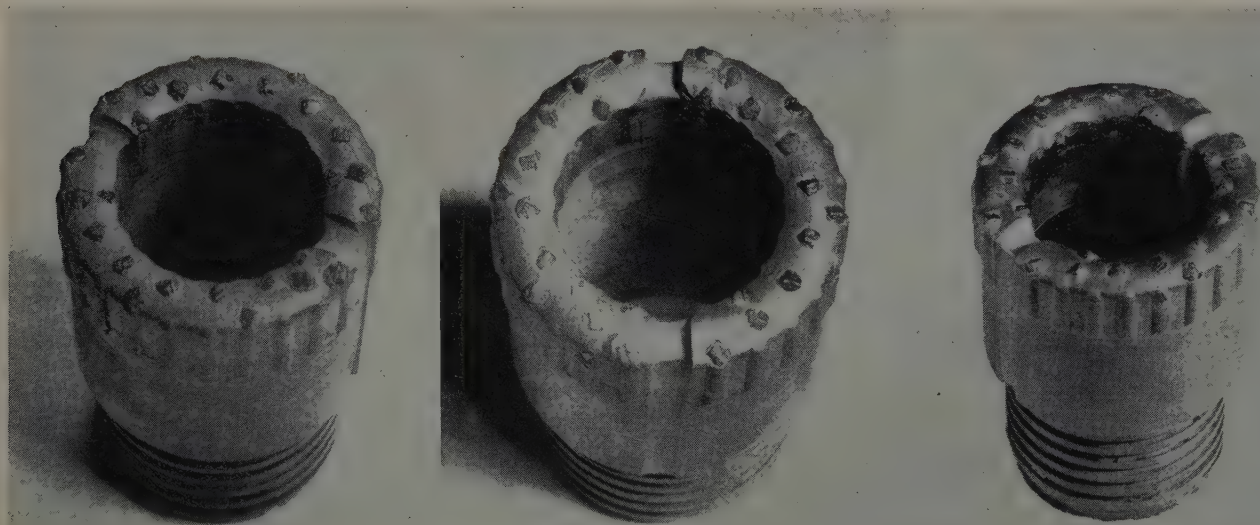


Fig. 1 (left)—Experimental drill bit M-15 having a point orientation with a positive (*gouge*) rake off cube face. Apparent rotation, counter clockwise. Fig. 2 (center)—Bit M-14 having a point orientation and a negative (*drag*) rake off cube face. Rotation counter clockwise. Fig. 3 (right)—Bit M-16 having edge orientation and a positive rake (20° to 25° *gouge* angle) off cube face. Apparent rotation counter clockwise.

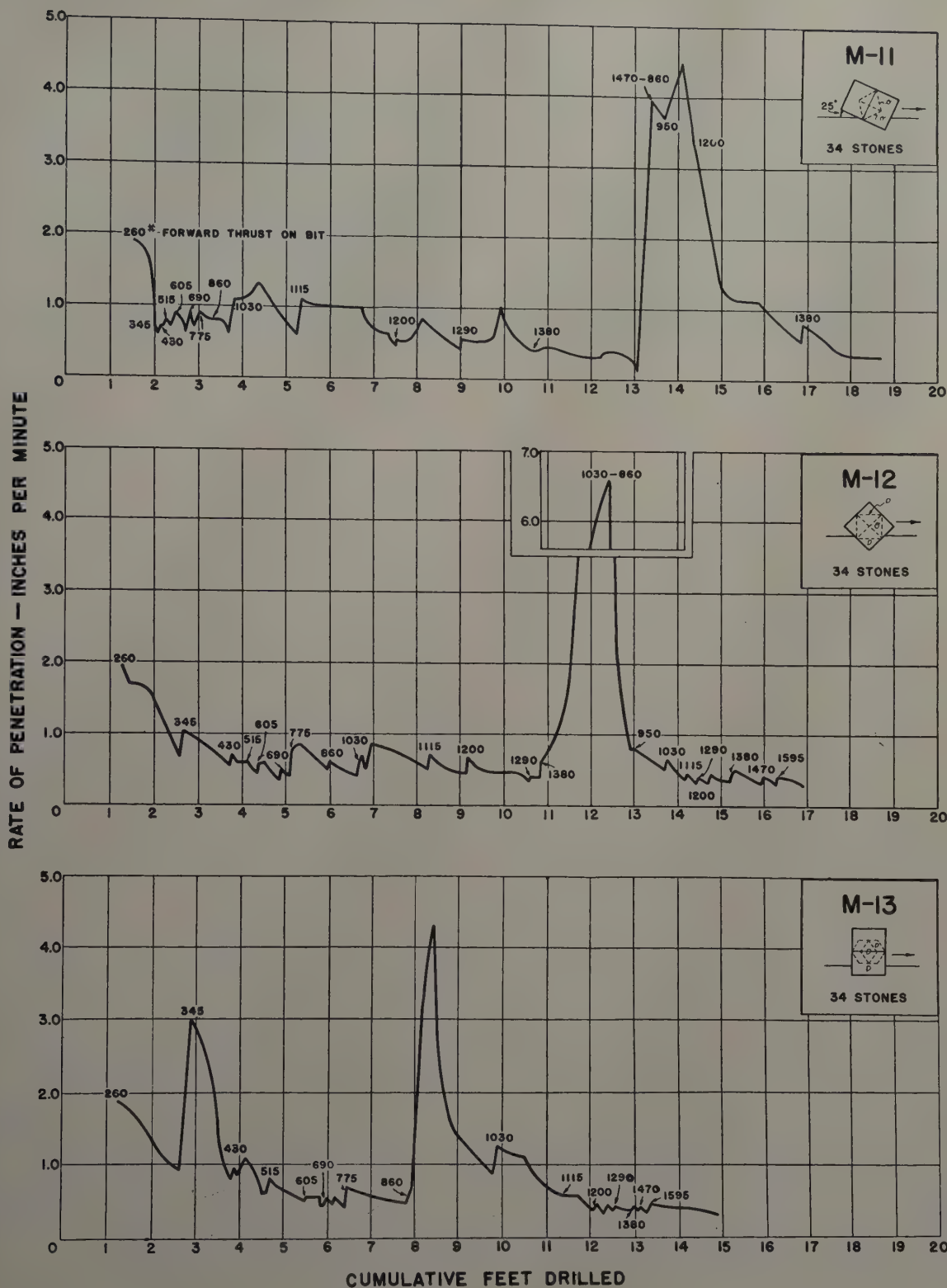


Fig. 4—Test history of EXT diamond drill bits M-11, M-12, and M-13. Constants: 1000 rpm of drill and 2 gpm of water. Variables: Forward thrust of bit and rate of penetration.

Five different orientations were tested:

1—A positive rake off the cube face, with the cube corner or point leading. This gives a negative rake off the octahedral direction, which was found to be a preferable orientation when an octahedral diamond

was used. Bits M-11 and M-15, Fig. 1, had this orientation.

2—A negative rake off the cube face with the cube corner or point trailing. This gives a positive rake off the octahedral direction, considered an in-



Fig. 5—Bit M-11 (X4) having point orientation and a positive (gouge) rake, showing formation of abraded flats after 10.9 ft drilling in granite. Rotation counter clockwise.



Fig. 6—Bit M-15 (X8) having point orientation and a positive (gouge) rake, showing gage stone fractured along octahedral cleavage after 13.0 ft drilling in granite.

ferior orientation with an octahedron crystal.* See

* The random orientation by a bit setter of a cube diamond in the round depressions of a standard bit mold would be normally an on point orientation similar to No. 1 and 2.

bit M-14, Fig. 2.

3—On edge of cube, with edge normal to direction of cutting. Contact then is parallel to the hard direction of a dodecahedral face. An example is bit M-12.

4—On edge of cube, with edge parallel to direction of cutting and hence comparable to drilling along soft direction on a dodecahedral face. An example of this is bit M-13.

5—On edge of cube with a positive (*gouge*) rake off the cube face, equivalent to a negative rake off the hard direction of a dodecahedral face. Examples of this are bit M-16, Fig. 3, and bit M-17.

The only previous known testing of bits having

oriented cube diamonds was done in South Africa late in 1948.² In conjunction with the Rhokana Copper Corp., the Diamond Research Laboratory tested two EX non-coring bits having cube diamonds, 9 stones per carat, oriented so they would drill on the octahedral plane. These would be on-point orientations. Their bit No. 1, corresponding approximately to bit M-14 described in this paper, drilled a total of 197 ft, whereas bit No. 2, comparable to orientation in bit M-11, Fig. 4, drilled 349 ft. Average footage of a normal production bit of comparable caratage was stated to be only 150. The Rhokana Copper Corp. was enthusiastic about the results and suggested an orientation of the cube diamond on the dodecahedral face, similar to bit M-12, to eliminate the tendency of cube corners to shear off along the octahedral cleavage planes.

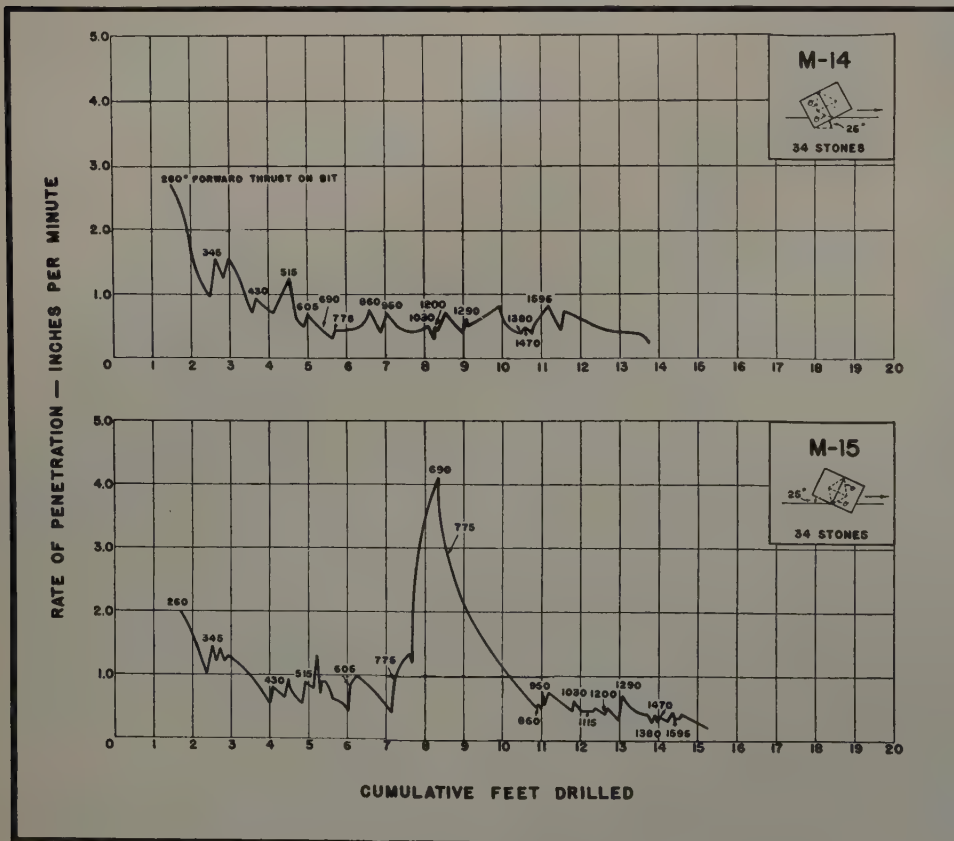


Fig. 7—Test history of EXT diamond drill bits M-14 and M-15. Constants: 1000 rpm of drill and 2 gpm of water. Variables: forward thrust of bit and rate of penetration.

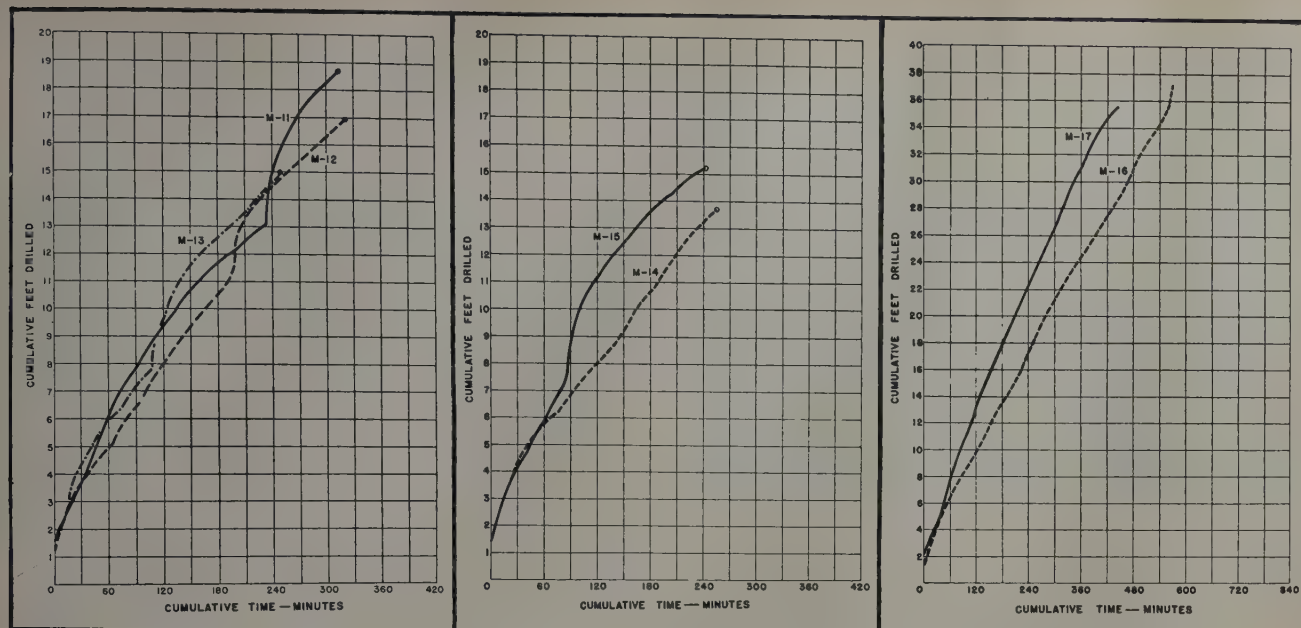


Fig. 8—Cumulative test data, bits M-11 through M-17.

Bits M-11 and M-15 both had point orientation, Fig. 1, with a positive or *gouge* rake off the cube face; hence they should be considered together. This orientation gives a resultant cutting force of diamond on rock that is more or less normal to the octahedral cleavage. Experience in the testing of octahedron diamonds showed this to be a preferential orientation. Such is not the case with cube diamonds. The points wore down rapidly and then tended to form polished flats, Fig. 5, with a resultant decrease in rate of penetration. When a critical pressure was reached, one or more of the diamond points failed, causing destruction of some of the gage diamonds, Fig. 6. This resulted in high, but short-life, advance rates between 13 to 15 ft for M-11, Fig. 4, and 8 to 10 ft for M-15, Fig. 7. Both bits gave approximately the same total footage, 18.7 and 15.3 ft respectively, and the same overall rate of advance, Fig. 8.

Bit M-12 had an on-edge orientation, with the edge normal to the direction of cutting. This orientation should be structurally strong, except for a weakness along the dodecahedral cleavage. In any event, there were indications of diamond breakage early in the life of the bit, and the drilling records showed erratic performance until the breakage became accentuated at about 11.5 ft. Abrasion and polishing of diamonds followed, with pronounced decrease in penetration rate. Overall performance, Figs. 4 and 8, was about the same as for bits M-11 and M-15.

Bit M-13 tested the on-edge orientation with the edge parallel to direction of motion. This is equivalent to drilling along the soft direction of a dodecahedral plane. The diamonds exhibited chipping and breakage in early life, giving periodic high rates of advance. Explanation for this might lie in the fact

Table I. Diamond Wear for Test Bits M-11 through M-17, Set with Oriented Cube Diamonds

Table I. Diamond Wear for Test Bits M-11 through M-17, Set with Oriented Cube Diamonds															
EXT Bits		M-11		M-12		M-13		M-14		M-15		M-16		M-17	
Diamond Type Size Number		Congo - Cubes 9 to 10 per carat 34 Stones (26 Crowns; 8 "Kickers")													
		Powdered Tungsten - Bonding Alloy													
Matrix															
Diamond Orientation and Exposure		+25° off Cube Face; -30° off Oct. Face 		45° off Cube Face; On Dodec. Face 		On edge of Cube; On Dodec. Face 		-25° off Cube Face; +30° off Oct. Face 		+25° off Cube Face; -30° off Oct. Face 		+22° off Cube Face; -22° off Dodec. Face 		Same as M-16	
Ft. drilled		18.68		16.95		14.93		13.72		15.27		37.20		35.62	
Diamond weights		Carats													
Original		3.76	3.76	3.76	3.76	3.48	3.48	3.50	3.50	3.58	3.58	3.50	3.50	3.53	3.53
Final		3.36		3.37		3.16		3.28		3.19		3.15		3.43	
Salvage (reusable)			3.07		2.99		3.07		3.21		2.72		2.95		3.43
Diamond loss		Carats													
Net from wear		0.40		0.39		0.32		0.22		0.39		0.35		0.10	
Total--after salvage			0.69		0.77		0.41		0.29		0.86		0.55		0.10
		Carats per Ft.													
Net from wear		.0213		.0230		.0214		.0161		.0255		.0094		.0028	
Total--after salvage			.0369		.0453		.0274		.0212		.0562		.0148		.0028



Fig. 9—Bit M-14 (X8) with 30° positive rake off octahedral plane, showing breakage of diamonds along cleavage planes in first foot of drilling during break-in period.

that an edge was sweeping around a circle of small radius, causing torsional forces to act along the edge. Following the formation of sufficient flats by diamond chipping, subsequent abrasion and polishing caused rapid fall off in bit performance. Drilling characteristics and advance were remarkably similar to M-15.

Bit M-14 had diamonds with a point orientation and the cube corner trailing, Fig. 2, and therefore has a positive rake off the octahedral cleavage plane. The corners of the stones started to cleave off in the first foot of break-in drilling, Fig. 9. This confirms previous experience with similar orientation of the

octahedron diamonds in bits M-2 and M-4. The tendency for the trailing edge of the stones to cleave away, followed by the formation of polished flats ahead of the broken edge, resulted in a rapid drop in bit performance, Fig. 4, and overall advance, Fig. 8.

Bit M-16, Fig. 4, and M-17 were set with the stones on edge, with the edge itself parallel to the radii drawn from the center of the bit. This is somewhat similar to M-12 with the important exception that the diamond was oriented with a 22° negative rake off the dodecahedral plane. This, then, gives a positive, or *gouge*, rake off the cube face, resulting in a strongly backed orientation of fast drilling characteristics. Thus at one time there is achieved a long cutting edge, a structurally strong orientation, and a gouging or planing action in drilling, all coupled with a diamond wear along the hard direction off the dodecahedral face. The test results bear out these contentions in a striking manner. Both bits made more than double the advance of the others in the group of seven, totaling 37.2 and 35.6 ft respectively in comparison to a range of 13.7 to 18.7 ft for the other five bits. Drilling performance was remarkably uniform, Fig. 10, except for a high rate of advance near the end of runs of M-16, when a diamond broke and temporarily roughened the others. The typical test history was one of slow gradual abrasion or chipping of the exposed edges, Fig. 11, with very little fracturing or breakage of the individual diamonds, even though the cubes had many points of incipient weakness such as fine hair-like cracks. Eventually the edges either abraded or chipped, Fig. 12, to a degree at which increased sur-

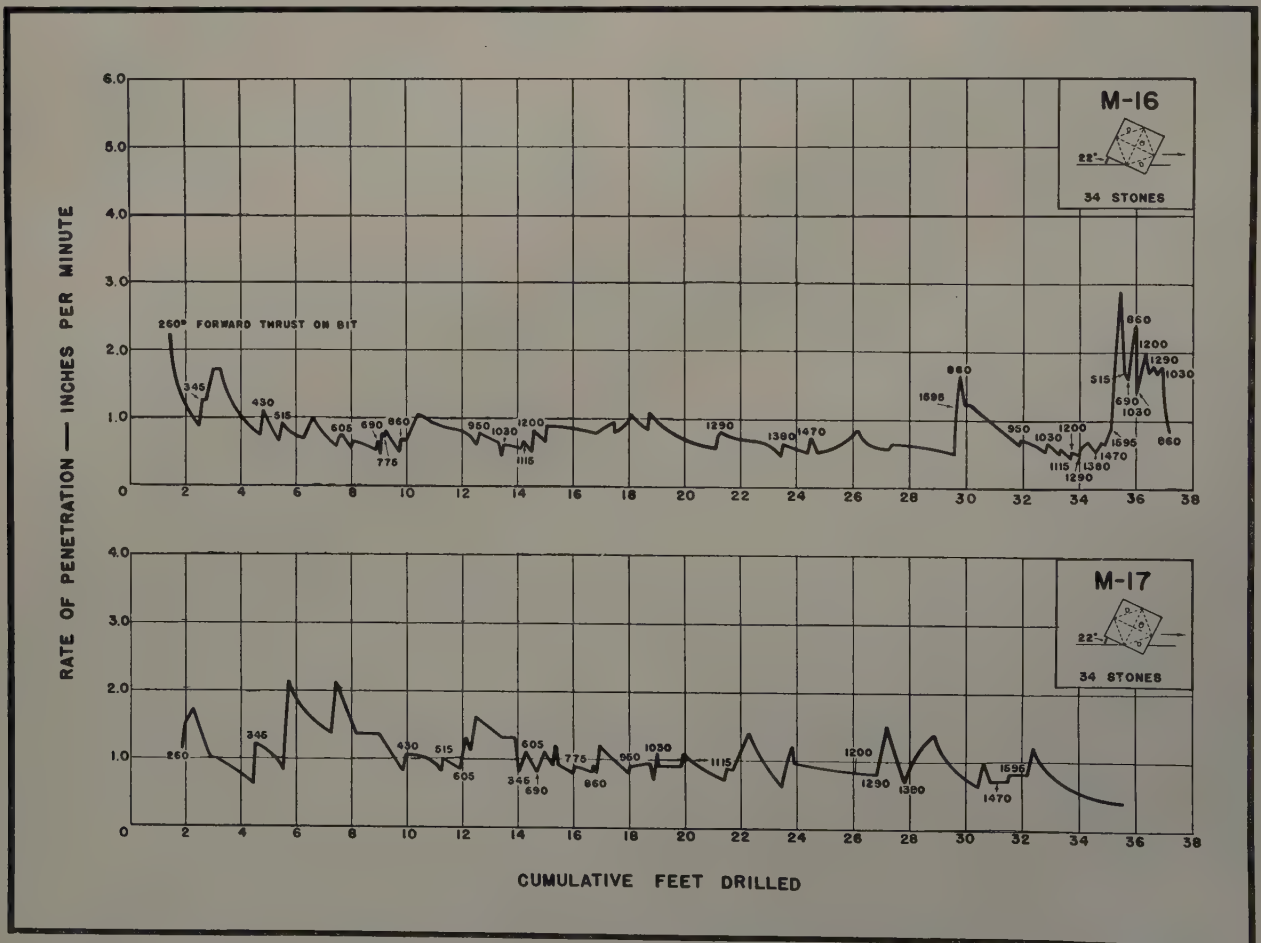


Fig. 10—Test history of EXT diamond drill bits M-16 and M-17. Constants: 1000 rpm of drill. Variables: forward thrust of bit and rate of penetration.

face contact would not permit appreciable penetration of individual diamonds to maintain the set minimum rate of 0.3 to 0.5 in. per min at maximum thrust of 1595 lb. Fig. 8 points up the similarity in performance of the two bits.

Diamond Wear and Loss

The objective of diamond orientation is to increase bit life and drilling rate and at the same time to decrease diamond wear. Often a compromise setting is required to accomplish these objectives, diamond loss being increased to maintain a good drilling advance. This is the case with the octahedral diamond. Such a compromise setting does not appear to be necessary for the cube-shaped stone.

Diamond wear for the group of seven test bits is presented in Table I. Bits M-11 through M-15 have somewhat comparative diamond loss as well as bit performance. Bits M-16 and M-17, with the greatly preferred orientation, show astoundingly low diamond loss. Had drilling stopped with M-16 at 35 ft, before a diamond broke and fractured others, the diamond loss for M-16 would have been approximately the same as for M-17.

All the diamonds from bits M-16 and M-17 were re-usable, and a big advantage of the cube shape is that there are eight similar edges for use in the re-setting process.

Comparison with Octahedron Diamonds

The test work has indicated that for drilling purposes the Congo industrial diamond having a cube form is superior to the Congo octahedron diamond of comparable quality and size. This is true of the

Table II. Comparison of Test Results for Diamond Bits Having Oriented Congo Stones of Cube and Octahedron Shape

Item	Octahedron Diamonds, Bits M-1 Through M-9	Cube Diamonds, Bits M-11 Through M-17
Minimum footage drilled per bit	4.0	13.7
Maximum footage drilled per bit	21.5	37.2
Average footage drilled per bit	12.1	21.8
Average drilling rate, all bits, in. per min		
Average drilling rate, bits with preferred orientation, in. per min	0.5	0.65
	0.9	1.2
Maximum diamond loss, carats per ft	0.086	0.056
Minimum diamond loss, carats per ft	0.012	0.003
Diamond loss, bits with preferred orientation, carats per ft	0.030	0.009



Fig. 11—Bit M-16 (X6) having edge orientation and positive rake (20° to 25° gouge angle) showing slight wear except for abrasion of inside gage stone after 16.8 ft of drilling in granite.



Fig. 12—Bit M-17 (X4) having edge orientation and positive rake (20° to 25° gouge angle) showing limited abrasion and wear after 20.0 ft of drilling in granite.

drilling rate, total footage, and diamond loss, since the strong structural shape of the cube presents well supported edges, at 90° angles, for cutting. In contrast, the octahedron either has points with acute angles of 70° that are weak and break easily or edges of wide angles, 110°, that do not easily penetrate the rock. Comparative results shown in Table II highlight the superiority of the cubes.

Summary

This phase of a research program testing the merits of diamond orientation showed the following results pertaining to the cube-shaped industrial Congo diamond.

1—It is superior to the octahedron, giving a longer bit life at a reduced diamond loss.

2—The cube is a structurally strong stone lending itself to an edge orientation that is fast cutting and of long life.

3—At preferred orientation the edge is set normal to the direction of cutting (parallel to the radii) and has a 20° to 25° positive, or *gouge*, rake off the cube face. This gives a 20° to 25° negative rake off the hard direction of the dodecahedral plane. Diamond loss can be astoundingly low, and the stone presents eight different edges for re-setting.

Acknowledgments

The author wishes to thank the Graduate School of the University of Minnesota for providing the funds to make this research possible and the J. K. Smit Co. for manufacturing the test bits gratis. The officials of Region V and the Mount Weather Station, Region VIII, of the Bureau of Mines and Mr. Robert Longyear of the E. J. Longyear Co. have been most helpful in furthering the diamond drill research projects. The staff and research assistants of the School of Mines and Metallurgy aided in the test work and in preparation of the manuscript.

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Fine Coal Drying

by G. A. Vissac

The drying of fine coal involves special techniques, which are discussed and analyzed. Types of dryers employing these techniques are described. Calculations are presented for new methods of dealing with the entrained dust that is always present in fine coal drying operations.

NEW conditions, new requirements, and new methods have increased the demand for more efficient and more economical methods of drying fine coal. Dewatering of larger sizes may reduce the surface moisture to 8 or 9 pct. It is more difficult, however, to dewater sizes below $\frac{1}{4}$ in., and some filter cakes still contain as much as 20 or 25 pct moisture. Increased freight rates and stricter consumer specifications have resulted in a demand for further reductions in moisture content. This can be obtained only by heat drying.

Most modern methods of heat drying disperse or spread the mass of coal to be dried, in an atmosphere of dry hot gases. The more intimate the contact between coal particles and hot gases, the quicker and more efficient the drying operation will be. Two different techniques are generally employed, using either a fluidized condition or an entrained condition of the coal to be dried.

Fluidized Condition

Fluidization of a body of sand was defined and explained by Fraser and Yancey in a paper published in 1926.¹ This condition was artificially obtained and maintained by proper regulation of the rate of air flowing through the sand body. "The sand bath 'boils' uniformly on the surface," they write, "and feels like a fluid." The fluidization technique was also described and analyzed by Steinmetz² in connection with the operation of an air cleaning table. His main conclusions are as follows: "Fluidity is, for the particles involved, the possibility of motion with minimum friction. . . . Then fluidity requires the introduction of various forms of energy capable of neutralising frictions. Two solutions can be used—air and/or mechanical motions (such as the shaking motion of the carrying deck of the air table). The combination of mechanical and air energy will give the widest margins of size ratios and of bed thickness, translated in capacity per unit area of the carrying table." Richardson and Langston³ have indicated results obtained with a dryer working with a fluidized bed. They used a vertical tube type of dryer, however, without the assistance of any mechanical energy, and without any lateral motion of the fluidized bed. The capacity of such a dryer is too limited for practical applications, since the speed of the acceptable air currents is held to the speed of fall of the particles involved. Capacities as low

as 182 lb of coal per hr per sq ft of dryer area are indicated. As stated by Richardson: "A basic limitation to a fluidised bed dryer is that the velocities of the gas must be held within a definite range; with velocities of 10 ft per second, all coal minus 6 mesh in size will be entrained, and the operation is then similar to that of a Flash dryer."

A fluidized bed must be virtually static. The coal particles simply kept in suspension offer a minimum resistance to the flow of gases, insuring the most favorable conditions for rapid evaporation of surface moisture. However, very wet fine coal, i.e., over 12 pct of surface moisture, will be delivered in the forms of mud balls, or as a soggy, sticky mass, almost impossible to disperse, sticking and acting as a wet blanket on the deck. Strong currents of gases and wide deck perforations will be required to punch holes in the wet mass and gradually loosen and fluidize it. The mechanics of fluidizing a bed of coal in a gas medium for the purpose of obtaining the most efficient drying condition are entirely similar when the fluid used is water and the purpose is to break up and distend a bed of coal to be cleaned so that perfect stratification according to densities will be insured. Purely mechanical energy is used in the basket-type jig, water pulsations in the piston and in the Baum-type jigs. A combination of mechanical motion and of air pulsation offers the most efficient and favorable conditions.

Entrained Condition

The most critical factor to be considered in the design of a dryer employing the entrained condition technique is the speed of the hot gases to be circulated in the drying column. With insufficient gas velocity, excessive amounts of the largest sizes will drop to the bottom of the dryer column without being thoroughly dried. On the other hand, high gas velocity will cause degradation, dust losses, and high power consumption. Figs. 1 and 2, reproduced from Hanot,⁴ show the relative importance of speed and temperature for various sizes of particles. It can be seen, for instance, that to maintain in unstable equilibrium particles of $\frac{1}{4}$ -in. size in a gas current at 500°C, a speed of 30 meters per sec, or 6000 fpm, will be required. For $\frac{1}{2}$ -in. particles an almost prohibitive speed of 45 meters per sec, or 9000 fpm, will be necessary. In practice, maximum gas velocities of 3000 fpm are recommended; since power increases as the cube of the velocity, it can be seen that beyond certain limits such dryers would not be economical.

If the particles were moving at the same speed as the hot gases they would remain in the same layer of gases, which would quickly become sat-

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Discussion on this paper, TP 3568F, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, Dec. 1, 1952. Revised April 29, 1953. Los Angeles Meeting, February 1953.

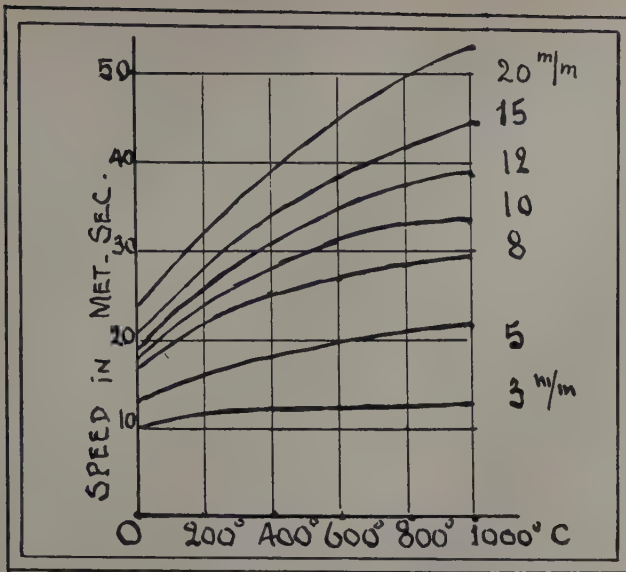


Fig. 1—Speed of gas flow required to float particles of a given size. (After Hanot, *Revue de l'Industrie Minerale*, July 1936.)

urated, limiting the drying possibilities. Accordingly, sliding motion must occur, and this increases with the size of the particle. Fig. 2 gives the relative speeds between gases and particles corresponding to a gas temperature equal to 500°C.

The above discussion clearly illustrates the actions occurring in the mass of coal entrained in a vertical dryer, and the resulting increased friction between particles. This is in fact the exact opposite of the condition in a fluidized bed in which frictions are reduced to a minimum.

Such are the discrepancies in published data dealing with the entrained technique that a simplified theory of the problems involved will be useful.* The

* In these calculations kilograms-meters-seconds are abbreviated as kgms. Speeds obtained in meters per second are also given in their equivalent values in feet per second.

weight of a cubical particle of coal of assumed density 1.25 and linear dimensions L , in meters, will be

$$W = 1250 \times L^3 \text{ kgms}$$

The total pressure on such a particle of an air current of speed equal to V will be

$$P = KL^2V^2 \text{ kgms}$$

K is a coefficient which will vary with the shape of the particle and the condition of the air. It expresses the weight of the mass of the atmospheric air column. Under normal conditions this weight is equal to 1.293 kgms and its mass to $1.293/g = 1/8$ (approximately), and with various conditions of pressure

and temperature $K = \frac{d}{8}$; d = density of actual air

or gases and $g = 9.81$ meters (acceleration of gravity). The equilibrium conditions for a particle falling in still air or for a still particle in moving air are

$$P = W \text{ or } 1250 L^3 = \frac{d}{8} L^2 V^2 \text{ or } V = \sqrt{10,000 \frac{L}{d}} \quad [1]$$

This is one of the fundamental equations. For example: $d = 1$, $L = 100$ mesh or 150 microns, and $V = \sqrt{1.5} = 1.24$ meters per sec, equivalent to 4.17 ft per sec.**

** Richardson³ gives 4 ft per sec.

For a temperature of 900°F, $d = 0.35$, and $V = 2.1$ meters per sec, or 6.88 ft per sec. For $L = 6$ mesh = 3 mm, the formula gives $V = 18$ ft per sec. According to Hanot, $V = 30$ ft per sec, and according to Richardson, $V = 10$ ft per sec.

Temperature of the Hot Gases

It is easier to dry coal at a high temperature, because the amount of moisture which can be absorbed by unit volume of gases is higher and the time required for a given heat transfer is shorter, but the use of high temperature involves many dangers, namely:

1—Oxidation of the coal, resulting in deterioration of its coking and burning properties.

2—Distillation of some of the hydrocarbons and production of explosive and obnoxious gases, as well as losses in heat values.

3—Losses in overall efficiency in heat balance, due to larger losses in radiation and in entrained heat in discharged or exhausted products.

This can be simply illustrated as follows. In a drying plant treating 200 tons per hr, delivering a dry coal at a temperature of 250°F with an outside temperature of 50°F, the heat losses will amount to $200 \times 2000 (250 - 50) 0.25$ (specific heat of coal) or 20,000,000 Btu's per hr. With a coal at 10,000 Btu's, there would be a total heat efficiency of 50 pct, a loss of 2 tons of fuel per hr. At a value of \$4 per ton this is a loss of \$8 per hr, \$160 per day, and \$40,000 per year.

4—Excessive speeds of the hot gases as their density decreases as indicated by the above formula for a given tonnage of coal of given sizes, resulting in greater degradation, higher dust losses, and higher power consumption. In short, everything else being equal, best results and efficiency require use of the lowest possible temperatures.

Vertical Dryers

Early dryers were mostly of the revolving drum type or of the scraper type in which a series of revolving scrapers cause the coal to drop from shelf to shelf against ascending currents of hot gases, Fig. 3. New developments have been using mostly vertical types of dryers in which the wet coal falls or cascades against a current of hot gases or is entrained by it.

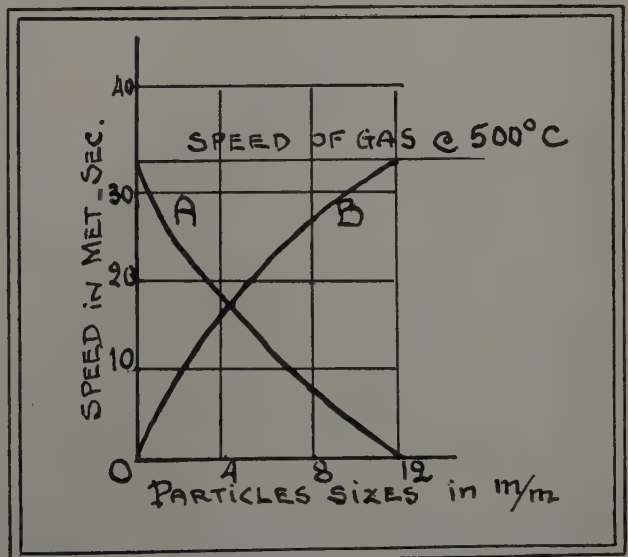


Fig. 2—Relative speeds of gases and particles. (After Hanot, *Revue de l'Industrie Minerale*, July 1936.)

In the tower type of dryer illustrated in Fig. 4 the wet coal is fed at the top and cascades by gravity over a series of adjustable inclined plates against a current of hot gases. The larger sizes are extracted by a screw conveyor at the bottom, and the finer particles are entrained and settled in a separator between the dryer and the exhaust fan. In other types of tower dryers star feeders instead of shelves are used, Fig. 5.

In the tube dryer the coal is fed at the bottom of a vertical pipe and entrained with the hot gases. Several types of this system are used in this country. This paper will discuss a well-known German construction, the rapid dryer or Rema-Rosin, see Fig. 6, manufactured by Buttner. The dryer has been described by Hanot.⁴

Hot gases from a pulverized fuel furnace are delivered at the bottom of a tube 33 ft high and 3.3 ft in diam. Wet coal is fed to the bottom of the tube from a surge bin through a screw conveyor and a spreader. Any escaped large piece or ball which might fall is crushed by a wheel located at the lower end of the elbow. Hot gases and entrained coal pass through a separator, the large particles or balls are returned in the feed zone, and the gases and lighter material are delivered to a bin where some of the

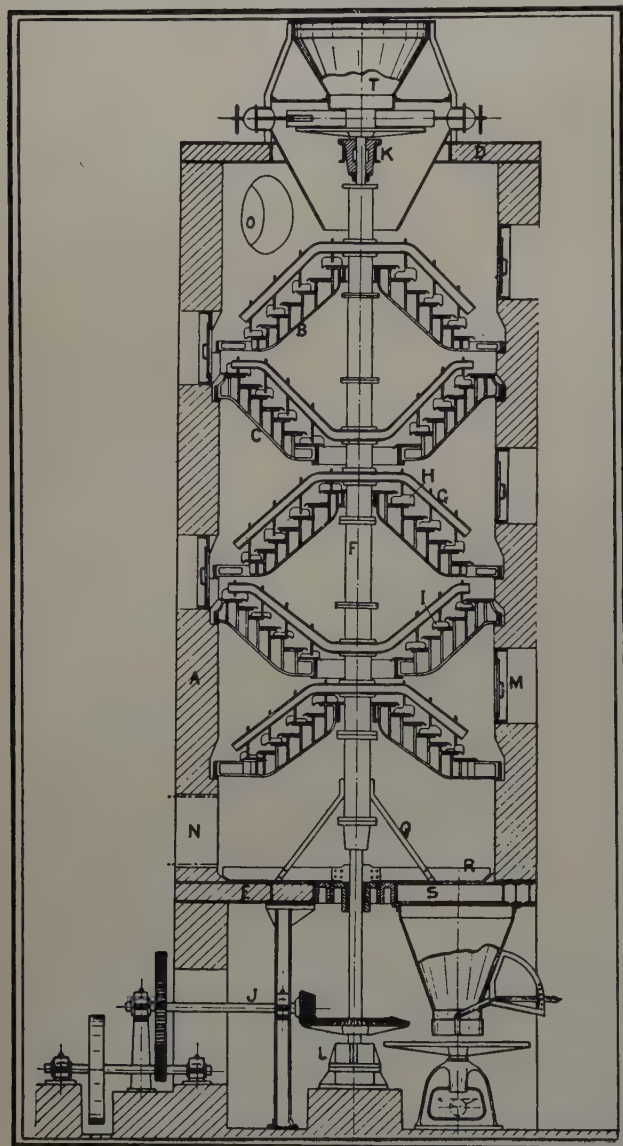


Fig. 3—Vertical dryer. Scraper type. Sahut, Conreur et Cie., France.

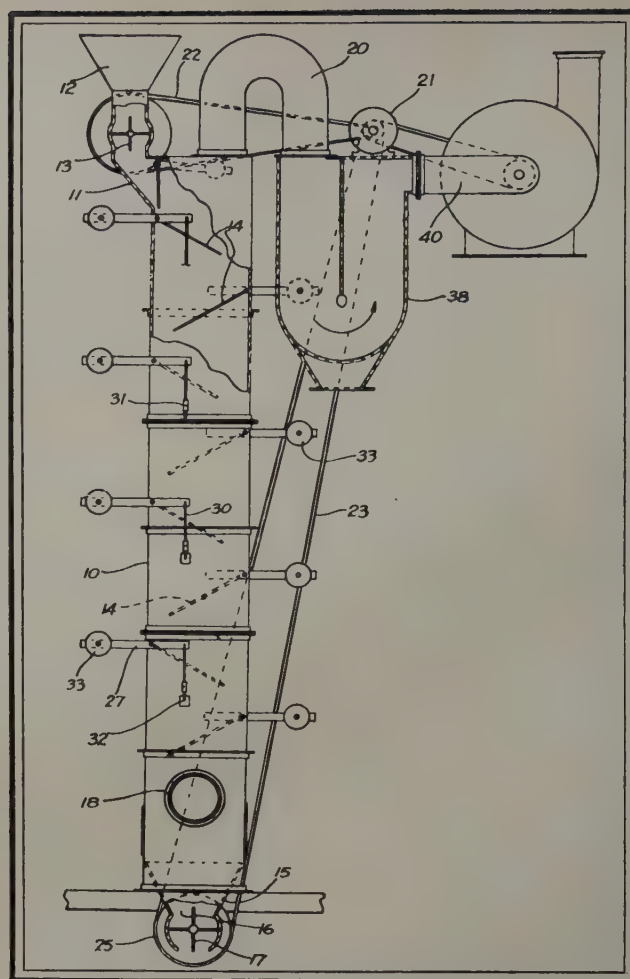


Fig. 4—Vertical dryer. Square tube, gravity type. U.S. Patent 2,317,003.

larger coal will settle. A cyclone collects some of the fine particles entrained, and the balance is settled in a washing tower on the discharge of the exhaust fans. On a bituminous coal containing 20 to 25 pct moisture, tests indicated outputs of from 16.7 to 31.8 tons per hr, with a dry product at 0.82 to 3.04 pct moisture. The fan motor had a rating of 150 kw.

However, in Europe very few of these dryers are used on bituminous coals. They treat mostly brown coals to remove inherent moisture. Fraser and Driessen⁵ report two such large installations for hydrogenating plants at Offleben and Lutzthendorf using seven Rema-Rosin dryers. Each dryer evaporates a ton of water per hour, a low figure compared to the American standard, and also an indication of one of the limitations of the entrained technique. When the drying process is to be followed by a chemical treatment excessive dust must be avoided, as it usually spoils some of the distillation products. In such cases lower gas speeds and therefore lower outputs are required.

The Pulso Dryer

The Pulso dryer, a screen type, is specially adapted to the drying of fine coals. Upward currents of hot gases are used instead of downdrafts used with the screen type for larger coal sizes. The coal is carried over a shaking screen, and fluidization of the bed is obtained by a combination of mechanical motions of the screen and pulsations of the currents of hot gases. Most of the coal is retained over the screen deck, but a variable proportion of the feed drops

through the screen to be dried in the hopper below against an ascending current of hot gases. Because of the perfect dispersion, this new patented method of drying the most difficult sizes under the deck^o is extremely efficient. Furthermore, this fine coal is trapped under the deck and extracted directly by a screw conveyor, without any exhaust gases, reducing difficulties of dust separation and collection.

Extensive tests and experiments conducted on this dryer at the Ernest mine of the Rochester and Pittsburgh Coal Co. have resulted in the use of a conditioning screen plate, an outstanding improvement. A feed containing a large proportion of fine sticky coals high in surface moisture acts as a wet blanket and is almost impossible to penetrate and fluidize. Stainless steel screen plates with round openings as large as 1 in. at the feeding zone of the dryer have greatly increased its efficiency.

A big advantage of the screen type of dryer with its truly fluidized bed is its capacity to handle a range of sizes up to 4 in. A condition similar to the feldspar bed used with some fine coal jigs is actually realized. The large pieces assist in opening up the bed periodically, and some of the heaviest fine particles, heavier because still overloaded with surface moisture, fall through the bed to be dried against the flow of hot gases in the hopper under the screen. With the entrained condition, as previously mentioned, the practical size limits are around $\frac{3}{8}$ in., but with the fluidized condition of the screen-type dryers there is almost no limit to the top sizes which can be treated. All sizes washed, including filter cakes, after whatever dewatering is economically possible, can be fed together to the dryer. Re-screening, if and when required, will be done under better conditions on the dried product, will be more

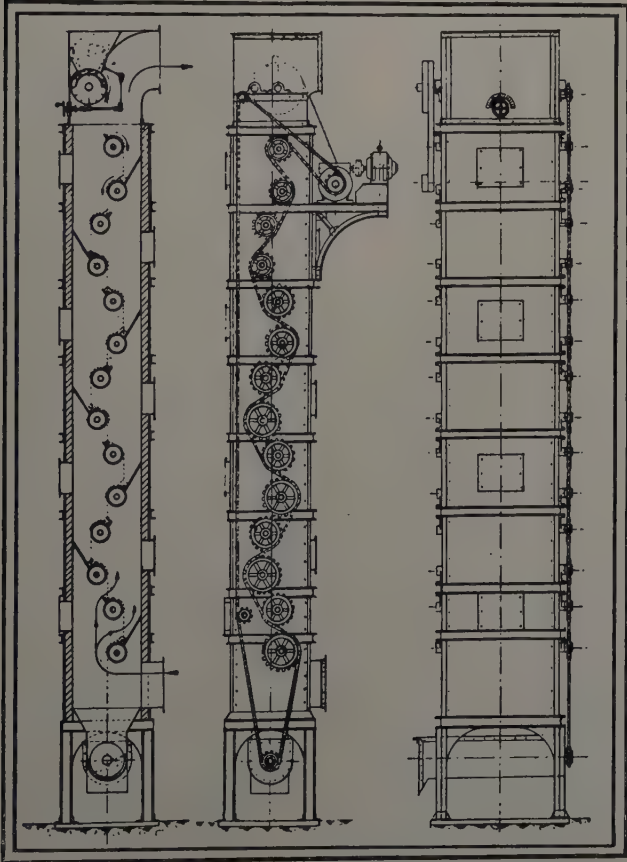


Fig. 5—Vertical dryer. Square tube and star feeders type. Sahut, Conreur et Cie., France.

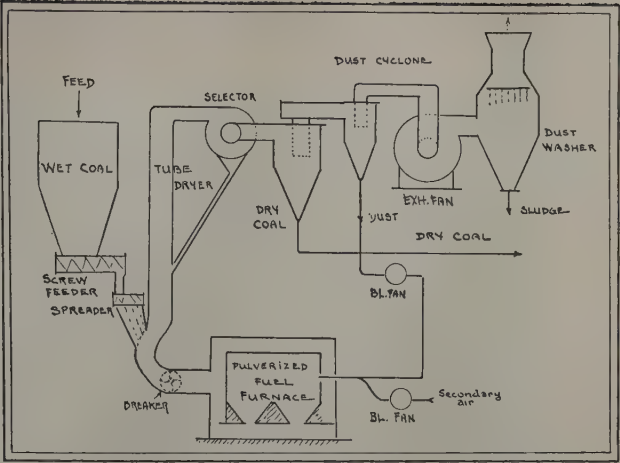


Fig. 6—Vertical dryer. Circular tube type. Rema-Rosin patents by Buttner. (After Hanot, *Revue de l'Industrie Minérale*, July 1936.)

effective, and will result in better appearance for the screen sizes.

Another great advantage offered by such a dryer is the lack of degradation, as indicated in Table I, resulting in fewer fines, less dust, fewer losses in handling and transportation, and, for many users, easier carbonization.

In earlier designs of this type, Fig. 7,⁷ with combined use of a blower fan at the furnace and a suction fan at the exhaust, it was found difficult to obtain proper fluidization. Blower fans were then introduced to inject tempering cold air under pressure. With a screening area of 200 sq ft the capacity was only 25 to 30 tons per hr of $-\frac{1}{4}$ in. coals.

Great improvements were obtained by reduction of the screening area to one single deck of 72 sq ft and the use of a larger blower fan between furnace and dryer. But with high drying temperatures this hot gas blower fan was considered a vulnerable feature. It was eliminated by the Rochester and Pittsburgh Co. tests. The new alternative system now

Table I. Rochester and Pittsburgh Coal Co. Ernest Mine Tests on Pulso Dryer

Item	Test G, June 13, 1952		Test L, Aug. 6, 1952	
	Feed	Product	Feed	Product
Feed	62 tons per hr at 10.8 pct moisture		46.2 tons per hr at 13.8 pct moisture	
Product	57 tons pr hr at 2.8 pct moisture		40.0 tons per hr at 2.6 pct moisture	
Evaporation	5.1 tons per hr H ₂ O		5.2 tons per hr H ₂ O	
Screen Analysis of Products				
Sizes	Pct	Pct	Pct	Pct
+ ½ in.	2.4	1.2	0.8	0.2
- ½ + ¼	43.9	43.2	1.9	1.2
- ¼ + ⅛	20.4	21.8	15.2	14.4
- ⅛ in. + 10 mesh	14.1	14	29.7	29
- 10 + 20	7.0	7	12.8	14.2
- 20 + 48	6.2	6.3	20.3	20.5
- 48 + 100	2.3	2.6	8.6	9.6
- 100 + 200	1.3	1.4	3.9	4.6
- 200	2.4	2.5	6.8	6.3

Average results with 10 dryers dealing with various coals and conditions indicate:

Feed	50 to 75 tons per hr
Evaporation	3 to 5.7 tons water per hr
Hot gas temperature	450° to 1150°F
Exhaust temperature	140° to 250°F
Discharged coal temperature	80° to 100°F

consists of one blower fan at the furnace and one exhaust fan at the outlet of the dryer. No moving parts are now left in the path of the hot gases, and temperatures of 1000°F or greater can be used, resulting in a larger output per unit area of the screen dryer up to almost a ton of wet coal per square foot.

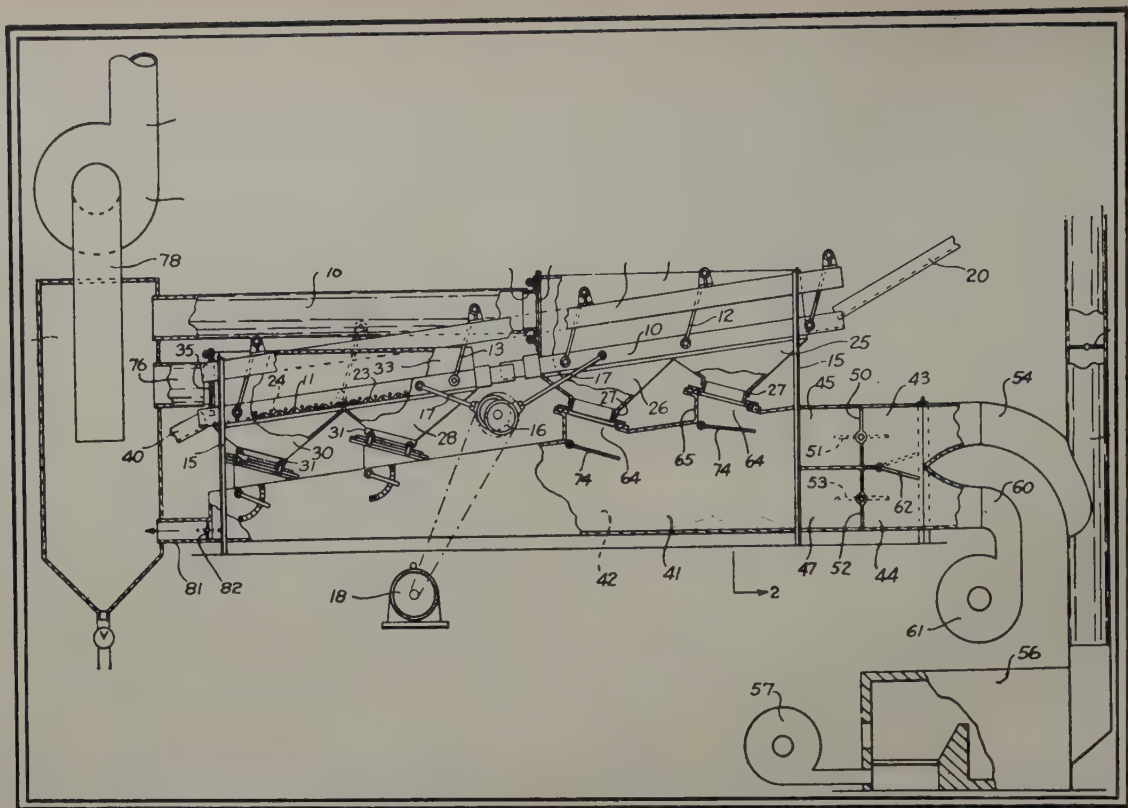


Fig. 7—Screen-type, updraft dryer. U. S. Patent 2,245,881.

In this new system now reduced to two fans, the required fan balance is easier to adjust, insuring minimum pressures at all vulnerable points and eliminating the dangers of fires or explosions, as well as the nuisance and dangers of gas leakages. Typical results of operation are given in Table I. Feeds of over 60 tons per hr of wet coal have been handled successfully, with actual evaporations of from 5 to 5.7 tons of water per hr for a screen area of 72 sq ft. New models with a screening area of 90 sq ft and the same overall dimensions are expected to evaporate 6.75 tons of water per hr. Feeds of fine coals with moisture contents over 14 pct have been handled successfully, without re-circulation.

Table I also indicates the size analysis of the products. Test L deals with very fine coal. Test G deals with a much coarser feed. It can be seen that almost no degradation occurred in the operation, although this is a very friable coal.

A full and detailed description of this type of dryer cannot be presented here. Many mechanical details, such as drives, suspensions, air chests, pulsators, baffles, louvres, ducts, and furnaces, have been gradually improved and developed during the last 20 years and have resulted in a well-ried and efficient drying machine. The new dryer, Fig. 8, uses principles of automatic control originally developed for the downdraft type, namely, automatic control of a variable volume of hot gases at a given and controlled temperature. A new method has also been developed, based on the use of a constant volume of hot gases at a variable temperature, which for some applications results in higher efficiencies and flexibility.

Drying Efficiencies

Although it is difficult to measure all the factors involved in the efficiency of a drying system, elementary laws of fan operation and of thermody-

namics can be applied with considerable accuracy. For a given drying problem, at a given temperature, all dryers must use the same weight of hot gases (unsaturated), but the most efficient operation will be accomplished by the system providing the most intimate and the longest contact between the hot gases and the coal particles. Generally speaking, the higher the drying temperature, the lower the heat efficiency, the higher the volume of gases for a given weight, and the lower the fan efficiency. The smaller the section of drying area, the more difficult the dust problems and the higher the resistance. Fan power increases as the square of the resistance.

In pressurized systems recirculation of a fraction of the exhaust gases may be required for safety, but it decreases efficiency. So many Btu's are introduced, but the same amount is withdrawn, and the heat balance is zero. Furthermore, it takes as much power to circulate a given weight of fresh unsaturated gases, but their drying capacity is much higher. D. R. Mitchell shows that "as the percentage of moisture in the drying gases increases, the rate of drying decreases."⁸ It may appear that the capacity of a drying unit can be increased simply by the increase of certain factors, or influences, involved, but this may result in nonpractical and inefficient solutions.

The following examples will illustrate methods of calculation used for a drying operation.

In all modern dryers, moisture is removed by absorption from hot gases and evaporation at temperatures under 220°F. Even with hot gases at 1000°F, average temperature of evaporation is 167°F; evaporation at 220°F would require gas temperature of over 5000°F. The fact that coal and exhaust gases are discharged at higher temperatures is only an indication of low efficiency.

A standard unit evaporates 5 tons per hr of moisture; corresponding tonnages of feed may be from

100 tons per hr at 5 pct moisture to 50 tons per hr at 10 pct. Assuming that a dryer treats 50 tons of wet coal per hr and removes 5 tons of moisture per hr, a fan will generally be required, working under 16-in. water gage and delivering 30,000 cfm, all reduced to 70°F and sea level. The corresponding size of motor required is $(30,000 \times 16)/4000 = 120$ hp. To double the capacity of the dryer (100 tons per hr feed and 10 tons per hr moisture) will require 60,000 cfm, but since water gage and power vary as the cube of the volume, the size of motor required for double the volume will be $120 \times 2^3 = 960$ hp. In a vertical dryer with a 3-ft diam drying chamber, a volume of 60,000 cfm becomes 180,000 cfm at 1130°F and 120,000 cfm at 600°F. The corresponding air velocities are 25,400 fpm and 17,000 fpm, or for a drying length of 25 ft a drying time of 6/100th to 8/100th sec.

Obviously two 50 ton per hr units will be a better solution than one 100 ton per hr unit.

Dust Collectors

The most difficult problem in operating all fine coal dryers is dust control. As large volumes of gases are used to obtain large drying capacities, it becomes more difficult to collect fine particles of dust produced in the drying of fine coals entrained by the hot gases and to prevent their escape in the atmosphere. The escape of these fine particles is a nuisance and a danger, as well as a loss in saleable coal. A loss of 2 pct in weight, not uncommon with certain types of dryers, on 100 tons per hr, 20 hr per day, 250 days per year, at a value of \$4 per ton, amounts to \$40,000 per year, or \$400,000 over a period of 10 years. Losses of such magnitude should obviously be considered in any competitive analysis of various types of dryers.

Since it is difficult to collect fine coal dust, the amount entrained in the exhaust gases must be held to a minimum. This may be accomplished by trap-

ping of some fine coals below the drying deck, by fluidization of the feed, and by absence of degradation. A certain amount of dust will always be entrained, usually when there are large tonnages and when extreme drying is required. Modern methods of dust control are fully described in a recent publication.⁹ In the case of the present dryer, however, for reasons of safety and economy the cyclone is the only practical tool. Simplified calculations are here presented for determining the number and sizes of cyclones required in each specific case.

Assume a particle of coal of cubical shape, of linear dimension L , density 1.25, entering a cyclone of radius R , with a propelled speed equal to V , entrained by a flow of gas traveling at a speed equal to v . The particle will be subjected to two main forces: a force centrifuge F in kilos, and an air pressure P kgs, such as

$$F = \frac{W}{g} \times \frac{V^2}{R} \quad P = \frac{1}{8} L^2 v^2$$

To achieve separation and settlement of the particle, it is necessary to have

$$F \geq P$$

or as $W = 1250 L^3$ and $g = 9.81$, approximately:

$$125 L^3 \times \frac{V^2}{R} \geq \frac{1}{8} L^2 V^2 \text{ or } 1000 \frac{V^2}{R} = v^2$$

A third force, G , or gravity, is practically negligible with the sizes of particles involved; for instance, for $V = 10$ meters per sec, $L =$ microns, and

$$G = \frac{1}{800} P$$

Assume a volume of air of Q cubic meters:

$$v = \frac{Q}{\pi R^2}$$

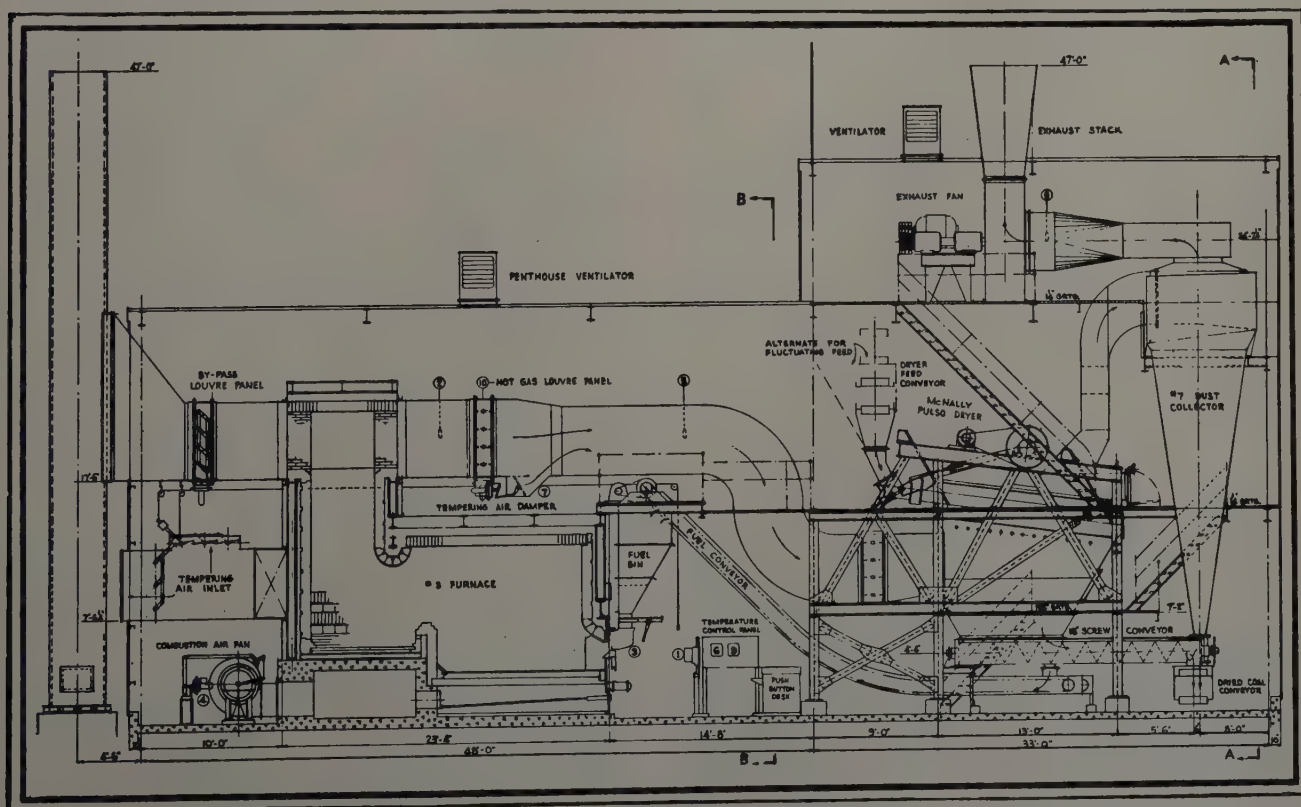


Fig. 8—Typical equipment arrangement of McNally Pulso dryer. Dual fan type. U.S. Patent 2,598,199.

the equation giving the conditions of settlement becomes

$$Q^2 \leq 10^4 \times V^2 \times R^3 \times L \quad [2]$$

It is generally assumed that the smaller the diameter the more efficient the cyclone will be. This is a consequence of the equation as given above of the centrifuge force F , in which the radius R of the cyclone enters as a divider. For instance, in the Van Tongeren patents in Europe, tubes as small as 5 in. diam are used, and an installation of dust collecting would contain as many as 680 tubes. As a result, air resistance and power consumption increase out of reason.

It has been found that 7x9-in. diam cyclones are best for the drying problems under consideration. The following examples are given for the above sizes according to the settling formula just presented. V is always taken at 20 meters per sec, or 3600 ft per min, the maximum allowable without undue vibration.

To settle all particles of a size $L = \frac{1}{10^5}$ or 10

microns or larger, in each cyclone it is necessary to use a volume equal to or inferior to Q , such as

$$Q^2 \leq 10^4 \times L \times V^2 \times R^3$$

$$\text{or with } L = \frac{1}{10^5}, V = 20, R = 1$$

$$Q^2 \leq 40 \text{ or } Q \leq 6.32 \text{ mcb per sec, or 13,000 cfm}$$

Accordingly, if a volume must be cleaned equal to 26,000 cfm, two No. 7 cyclones will be required. On the same basis, if an attempt were made to settle all particles down to 1 micron, the maximum volume allowable per No. 7 cyclone would be

$$2 \text{ mcb per sec or 4200 cfm}$$

and four No. 7 cyclones would be required for a total volume equal to 17,000 cfm. One No. 9 cyclone will efficiently handle almost twice the same volumes, for the same particles, sizes, or the same volumes down to half the size limit the No. 7 can handle.

In the above calculations, a minimum disturbance has been assumed. The required practical conditions are well known: 1—in the leading pipes a maximum speed of 3000 fpm; and 2—in the cyclone a maximum speed of 400 fpm for particles above 10 microns, 200 fpm for particles above 1 micron. Accordingly, the figure obtained from the above formula, on the basis of condition 1 must now be checked for condition 2.

$$V = \frac{Q}{\pi R^2} \text{ with } Q = 6, \pi = 3.14 R = 1$$

$$V = 2 \text{ m per sec} = 396 \text{ fpm}$$

$$Q = 2, \pi = 3.14 R = 1$$

$$V = 0.64 \text{ m per sec} = 130 \text{ fpm}$$

Condition 2 is also well in line.

In the above calculations, it has been assumed that the coal particle travels at the speed obtained at the cyclone entrance. However, in the case of tube dryers in which very high velocities are used, the speed of the particle will not be determined by the speed of the gases at the cyclone entrance, but because of its momentum will race ahead of the air current, just as a bullet races ahead of the propelling gases after leaving the muzzle of the gun. In such cases the momentum of the particle is the

dominating factor. It may cause the particle to shoot across the cyclone without a chance of being slowed down and collected.

Actual experience has shown the above formula to be sufficiently accurate to supply easy and convenient guides in the design of the cyclones required in normal coal drying operations. However, it must be pointed out that air currents must be kept within accepted limits to avoid complicated eddies whose motions are impossible to predict.

It is the author's opinion that the Stoke's Law used in some previous calculations, which applies truly to dense media and spherical particles, cannot be used in this case.

Design of Air Cyclones

An air cyclone designed by E. R. McMillan, Manager of the Northwestern Improvement Co. of Seattle, is particularly well shaped and designed for maximum efficiency. Most cyclones use a winding spiral, but the type and modulus of the spiral are very important, as are the length and angle of the bottom cone.

A pulsating flow used in these cyclones is also a new feature of great value,¹⁰ reducing the disturbing action of eddies and accelerating settling actions.

Economics of Fine Coal Drying

Assume that a dryer treats 60 tons per hr of wet coal, reducing the surface moisture content from 9 to 3 pct, or an evaporation of 6 pct or of 3.6 tons of water per hr. (This is in fact a low performance.) The number of Btu's required for the evaporation alone will be $3.6 \times 2000 \times 1000 = 7,200,000$ Btu's per hr. On the basis of an average overall efficiency of 50 pct, 1000 lb of coal per hr will be required at \$4 per ton. This gives the following operating cost:

Wages	\$2.00 per hr
Coal	2.00
Power	1.00 (100 kwh at 1¢)
Sundries	1.00
Total	\$6.00 per hr at 10¢ per ton of coal treated

The capital cost of such a plant installed will be around \$100,000. At 15 pct to cover depreciation and interest charges, there will be an annual cost of \$15,000 per year, or on an annual tonnage treated of 300,000 tons (20 hr per day, 250 days) or a capital charge of 5¢ per ton, a total cost of 15¢ per ton. As against the above charges, there will be a saving on freight and handling charges, assuming a total cost for these items of \$5.00 per ton, of 6 pct or 30¢ per ton, a credit balance of 15¢ per ton.

But in addition the use of dried coal will result in large savings in installation costs to the consumer. For example, in a coking plant with a unit cost of \$120,000 for an oven with a capacity of 18 tons in 18 hr, a reduction of 3 pct in the moisture content may cut the coking time by 50 min, an increased capacity of 5 pct, worth \$6000 per oven or \$600,000 for a total of 100 ovens (2700 tons per day plant). For a steam power plant with a capacity of 120,000 the capital cost may be around \$12,000,000. A moisture reduction of 6 pct will mean an increased capacity of 6 pct, a reduction in capital investment of \$720,000.

The figures and opinions presented above, in connection with the influence of heat drying on the coke oven technique, are confirmed and enlarged in

B. Hofmeister's paper,¹¹ just received, from which the following statements are quoted:

"A low moisture slack will run out faster and loading time can be reduced from 3 minutes to half a minute."

"Coking time can be reduced from 20-22 hours for a slack at 10% moisture, down to 16 hours, for a slack at 4%."

"For a coke plant with a capacity of 1000 tons, a 4% reduction in moisture results in a 25% increased capacity, a saving of .37 mark/ton."

"A decreased consumption of heat results in gas economies of 30 K. Cal/Kg, a saving of .37 mark/ton."

"A longer life for the brick walls results in a further saving of .10 mark/ton."

It is claimed that the greatly reduced loading time, with drier slacks, eliminates danger of explosions.

Desliming Vs Heat Drying

In the case of filter cakes at 20 to 25 pct of surface moisture, increased mechanical dewatering can be obtained by use of additional amounts of clean rinsing water. As much as an additional 2 pct of moisture can be removed in this manner.

Economically speaking, as the slimes removed have an ash content of 25 pct, and as the ash content of the product may have been improved by 1 pct, the corresponding additional loss in the coal treated would be 4 pct in weight. Based on the assumption that coal is \$4 per ton, the following balance sheet is presented:

Loss in weight	16¢
Gain in heat value:	
From 1 pct less ash	4¢
From 2 pct less water	8¢
Net loss	4¢

However, on the same basis the balance sheet of a heat drying operation would be:

Gain in heat value from 2 pct less water	8¢
Cost of drying, at 2¢ per ton per 1 pct water	4¢
Evaporation of 6 pct at 10¢ per ton	
Net profit	4¢

This may be the outstanding factor. Increased desliming may not remove more than 2 pct out of the 20 or 25 pct retained by a filter or centrifuge product, and in many cases this may not be enough.

Summary

This paper does not attempt a complete presentation of all types of dryers suitable for efficient drying of fine coals. The writer deals only with dryers he has had an opportunity to study and develop. He has examined the most important problems and presented some simplified methods of calculating the chief factors involved in the hope that it will contribute to further improvements in the techniques of fine coal drying and dust control.

Acknowledgments

The author wishes to express his thanks and appreciation to the staffs of the Riverside Iron and Engineering Works Ltd., Calgary; the Northwestern Improvement Co., Seattle; the Rochester and Pittsburgh Coal Co., Indiana, Pa.; and the McNally-Pittsburg Manufacturing Corp. for their contributions to the successful development of many vital improvements.

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Germanium and Other Elements in Coal And the Possibility of Their Recovery

by A. J. W. Headlee

RECENT interest in germanium centers around its use as a semi-conductor of electricity in electronic devices. It is generally believed that germanium transistors, diodes, triodes, photocells, and rectifiers will largely replace the present-day vacuum tubes. The germanium electronic devices

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Discussion on this paper, TP 3661A, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, Jan. 13, 1953. Joint Meeting, Central Appalachian Section, AIME, and West Virginia Coal Mining Institute, Huntington, W. Va., June 20, 1952.

are much smaller, lighter, and more resistant to shock, and they require only a small fraction of current required for conventional devices. Hence only a small amount of heat is given off by the equipment. This development will require many times the quantity of germanium now being produced, which in 1948 was approximately 1000 lb¹ and was obtained almost entirely from zinc ores in Missouri, Kansas, and Oklahoma. No one knows with certainty what the demand will be for germanium by 1956. It may be 40,000 lb.¹ Known concentrations in zinc ores are not sufficient to

maintain this yearly production of germanium, so that additional sources will have to be developed. Certain coals contain sufficient germanium to warrant their exploitation as a source.

Composition of Coal Ash

Recent researches² reported by Headlee and Hunter indicated that several other elements in addition to germanium may be recovered from coal ash. Before the occurrence of germanium in coal is discussed in detail in this paper, the overall composition of the mineral matter in coal will be presented.

Solid columns of coal 3 in. square were sawed from the full height of the seam and brought to the laboratory intact. Cubes of approximately 1 in. were cut at the top, every 3 in. thereafter, and at the bottom. These cubes were tested for porosity, for apparent specific gravity, and for mineral specific gravity. They were then ground to 60 mesh and analyzed for pyritic, sulphate, organic sulphur, and ash content. The ash was then analyzed spectroscopically for 38 elements.

An average analysis of ash for 596 cubes from 35 columns representing 16 coal seams is given in Table I. These coal seams include Redstone (columns 4, 10, 15, and 18), Pittsburgh (columns 5, 7, 8, 9, 11, 16, 17, 21, and 28), Bakerstown (columns 2, 14, and 20), Upper Freeport (column 3), Kittanning (columns 6, 12, and 13), Winifrede (column 27), Cedar Grove (column 34), Campbell Creek (No. 2 Gas) (column 33), Eagle (column 26), Sewell (columns 19 and 23), Beckley (columns 25 and 32), Pocahontas No. 9 (column 30), Pocahontas No. 6 (column 31), and Pocahontas No. 3 (column 29). The map locations are shown in Figs. 1 and 2. The rank of these coals ranges from high volatile A to low volatile bituminous.

Table I. Average Composition of Coal Ash

Oxide	Pct	Oxide	Pct
Lithium oxide	0.08	Chromium trioxide	0.018
Sodium oxide	1.78	Cobalt oxide	0.012
Potassium oxide	1.60	Copper oxide	0.062
Rubidium oxide	0.03	Gallium oxide	0.022
Calcium oxide	2.70	Germanium dioxide	0.013
Strontium oxide	0.41	Lanthanum oxide	0.030
Barium oxide	0.21	Lead oxide	0.048
Magnesium oxide	0.99	Manganese oxide	0.046
Aluminum oxide	30.5	Mercuric oxide	0.013
Silicon dioxide	44.8	Molybdenum oxide	0.016
Iron oxide	14.1	Nickel oxide	0.046
Titanium dioxide	1.51	Phosphorus pentoxide	0.30
Arsenic trioxide	0.018	Silver oxide	0.0015
Antimony trioxide	0.005	Tin oxide	0.022
Beryllium oxide	0.008	Tungsten oxide	0.012
Bismuth oxide	0.002	Vanadium pentoxide	0.050
Boric oxide	0.097	Zinc oxide	0.051
		Zirconium dioxide	0.029

Cadmium, cerium, and tellurium were looked for in each sample but were below the limits of detection, at 0.005, 0.01 and 0.05 pct respectively.

Sodium, potassium, rubidium, calcium, magnesium, silicon, chromium, and manganese are present in coal ash to a lesser extent than in the earth's crust. Aluminum, iron, titanium, beryllium, cobalt, copper, molybdenum, nickel, phosphorus, vanadium, and tungsten are 1 to 10 times as rich in coal ash as in the earth's crust, while lithium, strontium, barium, silver, arsenic, bismuth, boron, gallium, germanium, mercury, lanthanum, lead, tin, zinc, and zirconium are 10 to 185 times as concentrated in coal ash as in the earth's crust. Several of these 15 elements should be commercially recoverable from coal ash.

The above statements are based on an average for 35 columns. Some individual columns have values 2 to 5 times greater than the average so that ash in a certain area is richer in a particular element than the ash in another area.

Germanium in Coal

Germanium concentrations³ occur in the coal in a manner entirely different from that of any other element. These concentrations are shown in Table II.

Table II. Germanium Dioxide Content of Redstone Seam, Monongalia County

Depth, In.	Ash, Pct	Germanium Dioxide Content	
		Ash, Pct	PPM of "Ash-Free" Coal
1	14.7	0.017*	29
3	13.1	"	
6	24.4	"	
9	5.5	"	
12	4.8	"	
15	6.0	"	
18	7.3	"	
21	79.1	"	
24	54.1	"	
27	5.3	"	
30	5.3	"	
33	4.7	"	
36	23.1	"	
39	4.8	0.014	7
42	5.8	"	
45	14.8	"	
48	7.3	"	
51	13.7	0.021	33
54	28.0	0.033	141

* Below the limits of detection, 0.005 pct.

The germanium is concentrated in the top and bottom of the coal seam. In most cases the top and bottom inch of coal showed the highest concentration of germanium. The adjacent sample in each case is nearly 3 in. away. To find the vertical variation in greater detail, the top 3 in. of column No. 19, Sewell seam, and the bottom 4 in. of column

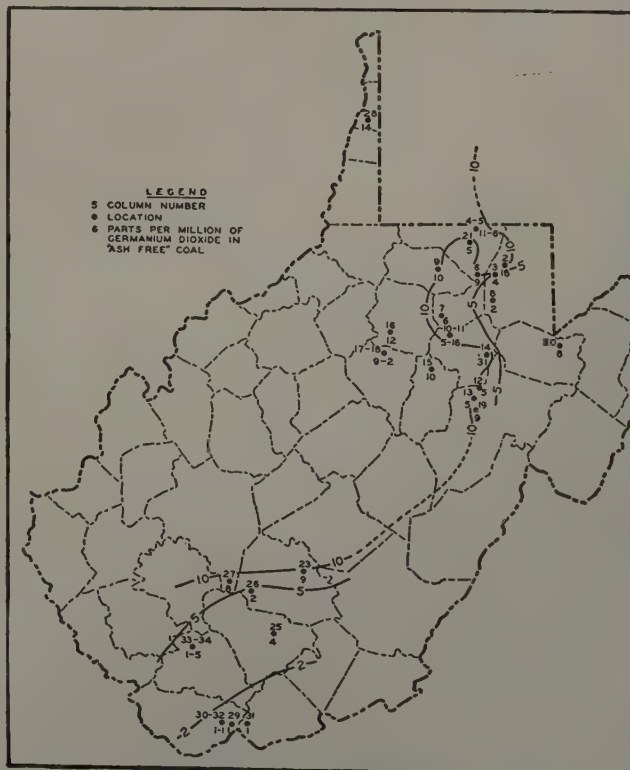


Fig. 1—Map showing isometric germanium dioxide lines for coal seams, full seam averages.

No. 2, Bakerstown seam, were sampled every ¼ in. and analyzed for germanium. These two sections were selected because of their high germanium content. The data are recorded in Tables III and IV.

The germanium content of the Sewell seam when calculated as percent ash is the highest in the fourth to eighth ¼-in. section and decreases from that high point. The germanium content when calculated as parts per million of ash-free coal is the highest in the upper ¼-in. of the top of the Sewell seam, while the third ¼-in is the highest in the bottom of the Bakerstown, with decreasing values above and below this. It would appear that the germanium was adsorbed by the coal as water entered it carrying germanium salts in solution. The other possibility is that the coal adsorbed germanium from gases containing volatile germanium compounds.

Screen and Float-Sink Tests on Bottom 3 In. of Bakerstown Seam

Twenty-four hundred pounds of the bottom 3 in. of the Bakerstown seam were obtained from a mine near Reedsville, Preston County, W. Va. The coal was screened and the various sizes were sampled and tested for ash and germanium dioxide content, see first part of Table V.

Table III. Germanium Content of Twelve ¼-In. Sections From the Top of Column 19, Sewell Seam

Section, ¼-In.	Ash, Pct	GeO ₂	
		Ash, Pct	PPM in Coal (Ash-Free)
First from top	8.49	0.13	121
Second	5.94	0.18	107
Third	7.08	0.11	84
Fourth	1.89	0.27	46
Fifth	2.20	0.24	54
Sixth	2.13	0.20	44
Seventh	2.25	0.22	51
Eighth	2.10	0.14	30
Ninth	1.89	0.08	15
Tenth	1.65	0.024	4
Eleventh	1.74	0.010	2
Twelfth	1.62	0.012	2

Table IV. Germanium Content of Fifteen ¼-In. Sections From the Bottom of Column No. 2, Bakerstown Seam

Section, ¼-In.	Ash, Pct	GeO ₂	
		Ash, Pct	PPM in Coal (Ash-Free)
Fifteenth	2.41	0.022	5
Fourteenth	4.75	0.008	4
Thirteenth	3.47	0.037	13
Twelfth	7.83	0.025	21
Eleventh	5.71	0.045	27
Tenth	4.00	0.08	33
Ninth	4.15	0.09	39
Eighth	2.79	0.25	72
Seventh	4.09	0.20	85
Sixth	4.78	0.19	96
Fifth	6.58	0.12	87
Fourth	8.99	0.10	99
Third	6.15	0.194	127
Second	6.15	0.083	54
First from floor	5.59	0.064	38

The screened fractions were washed with heavy media (magnetite) in a laboratory size washer. The float fractions were sampled and tested for ash and germanium dioxide content, second part of Table V. The shale refuse from the washer did not contain detectable quantities of germanium. It is to be noted that the germanium content of the ash is inversely proportional to the percent of ash

in the coal, and the germanium content when calculated on the "ash-free" coal basis remains nearly constant. It appears that the germanium is in the organic matter. The removal of as much of the ash as possible without losing coal prior to burning will concentrate the germanium in a much smaller quantity of the ash. Screening, then, does not separate the coal into high and low germanium coals according to particle size. However, those screenings which have the lowest ash content will have the highest germanium concentration in the ash.

Table V. Germanium Dioxide Content of Screened and Washed Fractions of Bakerstown Coal Sample

Screen Size, In.	Gravity	Bottom 3 In. Germanium Dioxide		
		Ash in Coal, Pct	Ash, Pct	PPM of "Ash-Free" Coal, Pct
+ 2		27.2	0.028	103
+ 1-2		40.4	0.015	102
+ ½-1		25.8	0.034	118
+ ¼-½		19.7	0.050	123
+ ⅛-¼		18.3	0.050	112
+ 1/16-⅛		13.0	0.076	114
- 1/16		16.8	0.054	109
(+ 2)	-1.34	9.3	0.10	103
(+ 1-2)	-1.30	10.0	0.10	111
(+ ½-1)	-1.34	8.5	0.10	93
(+ ¼-½)	-1.30	8.6	0.11	104
(+ ⅛-¼)	-1.29	6.8	0.13	95
(+ ⅛-¼)	-1.42	10.2	0.10	114
(+ 1/16-⅛)	-1.35	9.6	0.096	102
(+ 2)	-1.21	5.4	0.15	86
(+ 2)	-1.29	5.9	0.17	107
(+ 1-2)	-1.29	6.3	0.18	121

The concentration of germanium by screening depends on the particle size distribution of the shale. But screening is not a satisfactory way of separating mineral matter from coal. Washing technique does present an efficient method of removing mineral matter from coal, thus permitting the germanium to remain with a minimum amount of ash after coal combustion. Further static gravity separations are being made of the last 10 samples in Table V using potassium carbonate solutions of 1.28, 1.29, and 1.34.

Areal Variation in Germanium Content

The areal changes in germanium content of the coal are similar to those for volatile matter. These variations are shown in Fig. 1 for 35 samples from 16 seams.

Individual seams show a more consistent variation as shown for the Pittsburgh coals in Fig. 2. The germanium content of the coal appears to be proportional to the volatile matter. There is some indication that coals of lower rank than high volatile A bituminous coal decrease in germanium content.

Germanium in Ashes From Combustion Equipment

In the combustion of the coal varying amounts of the germanium are retained by the ash and may be concentrated under certain conditions. Extensive investigations were made in England⁴ in the past few years to determine whether or not these concentrations were sufficient to warrant recovery. Certain flue dusts in producer gas sets were found to be rich enough in germanium, about 0.5 pct, for its economic recovery. This dust collects in the colder portions of the flues conducting the gases away from the set if they are allowed to cool. If the hot gases are burned before cooling, germanium collects in the colder portions of the furnace flues. The total recovery from this source amounts to relatively small quantities of germanium.

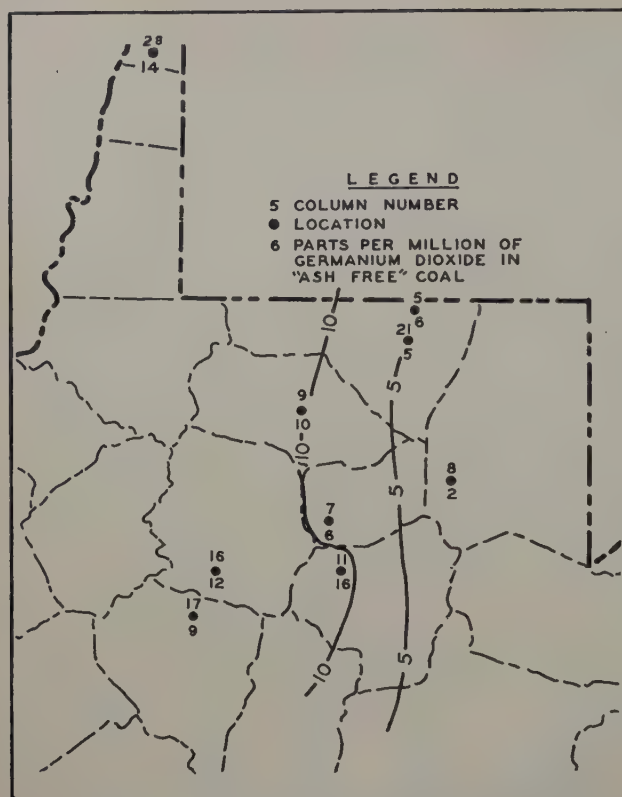


Fig. 2—The above map indicates the germanium dioxide in the Pittsburgh seam.

It does not appear likely that extensive recovery will be obtained except by the selection of low-ash highly germaniferous coals for specific combustion equipment in which a minimum of germanium is lost through the stack. Even in carefully controlled laboratory ashing operations some of the germanium is lost when the temperature never exceeds 740°C. It is not known with certainty just how much of the germanium is lost in combustion equipment. Estimates vary from a few to 70 pct of the germanium. However, when germanium recovery from coal ash becomes a reality, then compounds can be added to the coal to retain most of the germanium in the ash if necessary.

Methods of Recovery

Recovery of germanium from coal ash is long and tedious. This cost is reflected in the value of the finished product. Preparation of the germanium dioxide is further complicated by the fact that it must be of extremely high purity before it is suitable for use in electronic devices. A high percentage of the value of the product is required to offset the cost of purification. If it were necessary for common salt (sodium chloride) to be that pure it would probably cost a similar price per ton. Arsenic is particularly troublesome and difficult to remove. The final traces are removed by distillation of germanium tetrachloride through copper turnings. English^{1,4} methods of recovery from coal ash have been reported. American¹ methods have not been reported in detail. Methods of recovery will probably include the following steps: 1—fusion or its equivalent, 2—precipitation of the germanium as the sulphide, 3—conversion of sulphide to anhydrous chloride, 4—high efficiency distillation of chloride, 5—high efficiency distillation over copper turnings, 6—hydrolysis of chloride to germanium dioxide.

The end use of the germanium is as a metal. The dioxide is easily reduced to the metal with hydrogen above 700°C. Most of the electronic manufacturers prefer to produce their own metal from the oxide. Certain other metals have to be added in very small quantities to produce germanium metal with the desired electrical characteristics.

Economics

Germanium dioxide sells for \$142 per lb, or \$284,000 per ton. The price schedule for coal ash will probably be based on the germanium content of the ash. One prospective buyer has suggested that ash containing 0.2 pct germanium is worth \$75 per ton. Considerable tonnage of ash could now be prepared whose value might be \$25 or more per ton. Probably the minimum germanium concentration in coal ash having a recovery value is 0.05 pct. This lower limit will change as recovery research progresses and may go considerably lower. It is too early in this development to establish a firm price or lower limit.

Tests to date indicate that germanium will seldom be present in coal in sufficient quantities to mine and process coal for germanium recovery alone. Most of the mining cost will still have to be recovered by burning coal as a source of heat in conventional combustion equipment and recovering the germanium from the ash.

Conclusions

Germanium is commercially recoverable from coal ash. The bottom 3 in. of the coal seam can be selectively mined, washed, and burned to produce an ash rich in germanium. Certain coal utilization equipment concentrates germanium so that recovery is feasible from this source. Processes are available for recovery of germanium from the ash.^{4,6}

The recovery of germanium will be the incentive for the recovery of other elements from coal ash.

Acknowledgment

This paper is a review of the literature. The only unpublished data are those in Table V, regarding which the author wishes to express his appreciation to G. R. Spindler and H. W. Ahrenholtz, Jr., West Virginia University, School of Mines, for their aid and the use of their heavy media washer. Mr. Leslie Larsen, President, Fairfax Mining Co., very generously supplied the coal sample. The author gratefully acknowledges the aid of the staff of the West Virginia Geological Survey, and particularly that of Paul H. Price, for permission to publish this paper.

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The Application Of Centrifugal Forces To Gravitational Classifiers

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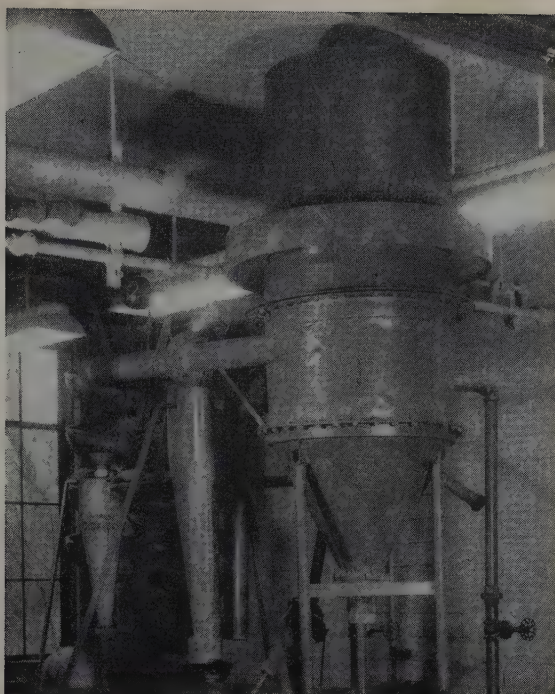
FOR many years gravitational classification has been employed as a basic tool in beneficiation of minerals and coal. While improvements have been made to increase efficiency and fields of application, the basic flow pattern has remained relatively unchanged in most gravitational classifiers. Feed slurry is introduced into a settling pool wherein the coarser solids settle against either an upward or horizontal fluid current, or a combination of both, and are withdrawn in a concentrated underflow stream. The finer solids below the classification point possess too low a settling velocity and thus are primarily swept out through a weir overflow by the bulk of the fluid. Because of its simplicity, efficiency, and low power and maintenance costs, gravitational classification finds a wide acceptance for separations in the range of 20 to 325 mesh.

Two factors resulting from the basic flow pattern employed in gravitational classification present difficult problems for certain installations. If classification at 60 to 200 mesh of separation is desired, capacity in terms of gallons of overflow per minute per square foot of settling pool area must be low to allow required solids to settle to the underflow. If a thousand or more gal per min of feed slurry must be handled, either a very large classifier or a number of smaller units must be employed, taking up floor space and head room. Examples of this are the large classifiers, wetting tanks, or settling tanks used in the coal industry for recovery of process water from fine solids. Units as large as 50 or 60 ft in diam and 60 ft high, requiring substantial amounts of steel, are not uncommon.

A second factor is the lack of *self-regulation* in gravitational classification. As overflow rate increases, mesh of separation increases correspondingly. This normally heightens difficulties of beneficiation. If a fine mesh of separation is required

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Discussion on this paper, TP 3612B, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, April 29, 1953. Revised May 15, 1953. Los Angeles Meeting, February 1953.



at varying feed rates, the problem is more than doubled in complexity.

One of the primary advantages of gravitational classification is low power requirement. Gravity feeds or very low pressure heads are employed. However, fluid energy is still present in the inlet stream and provision must be made to dissipate it properly to obtain uniform upward or horizontal currents. Rather than waste this energy, it would be desirable to utilize it to secure a greater degree of self-regulation and simultaneously increase classification capacity per square foot of area. A free vortex could be realized by a tangential entry of feed, resulting in generation of centrifugal force to complement force of gravity in its action. With greater force it should be possible to gain finer separation at any overflow rate per square foot of settling pool area as compared with conventional gravitational classification. Furthermore, as feed rate increases, centrifugal force will also increase and at least partially offset faster currents sweeping solid particles to the overflow. A more stable classification point should result.

The liquid-solid cyclone is the most common application of a free vortex in the beneficiation field.¹ Many previous papers have emphasized uniformity of flow pattern within the cyclone which is largely responsible for the sharp classification that can be obtained.²⁻⁴ If a free vortex can be maintained in a non-pressurized vessel with a free surface it may be possible to realize a third advantage of sharper classification. This open-top cyclone would then achieve a minimum energy loss comparable to that with conventional classifiers, and a mesh of separation coarser than 150 mesh, extending the range of application of the free vortex. This is not presently possible with ordinary cyclones.

To test the applicability of the free vortex to gravitational classification, a 30-in. diameter open-top cyclone was constructed at Northwestern University. The cylindrical section was 30 in. high and the conical section included an angle of 60°. The feed nozzle was a standard 4-in. pipe entering tangentially, with centerline 17½ in. below the top of the cyclone. The conical section was flanged near

Table I. Results for Open-Top Cyclone at Varying Feed Rates and Volume Splits, All Solids 2.83 Sp Gr Dolomite

	Run 113	Run 114	Run 115	Run 116	Run 117	Run 118	*Run 119	Run 120
Slurry Rate, GPM								
Feed	138.2	142.9	157.6	85.5	209.3	159.0	162.3	165.7
Overflow	117.0	121.0	132.0	71.0	183.2	129.1	148.8	127.2
Underflow	21.2	21.9	25.6	14.5	26.1	29.9	13.5	38.5
Solids Concentration, Wt, Pct								
Feed	22.6	23.4	24.5	23.5	22.6	22.0	19.10	22.0
Overflow	9.06	9.16	9.31	9.2	9.00	9.60	9.74	9.03
Underflow	66.1	69.0	69.4	66.0	74.5	57.6	75.7	52.0
Solids Rate, Tons per Hr								
Feed	9.08	9.80	11.42	5.91	13.87	10.12	8.81	10.56
Overflow	2.79	2.92	3.26	1.73	4.37	3.28	3.85	3.04
Underflow	6.29	6.88	8.16	4.18	9.50	6.84	4.96	7.52
Recovery in Underflow, Wt, Pct								
+20 mesh, U.S. standard	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
20x40	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
40x60	99.7	99.8	99.7	99.9	99.7	99.7	99.8	99.9
60x100	96.6	96.2	96.2	99.3	93.7	95.7	93.2	95.8
100x140	80.4	80.3	81.7	87.8	68.3	74.2	53.0	77.1
140x200	45.4	52.6	56.2	61.1	42.8	46.0	19.9	45.5
-200	6.0	9.3	9.9	12.4	7.7	12.6	4.3	17.0
Diameter of Separation, Microns	138	131	142	111	149	145	141	145
Equilibrium Particle, Microns	99	94	92	88	108	104	129	104
Classification Efficiency at 100 Mesh, Pct	83.1	81.0	84.6	76.9	84.1	82.9	91.5	78.9
Liquid Level above Bottom of Horizontal Overflow Section, In.	11	12	17	4.5	28	17	17	

* Overloaded underflow discharge.

the apex to permit insertion of different 60° underflow cones with outlet diameters ranging from 5/8 to 1 3/4 in.

Four-inch copper tubing was used for the overflow, the inlet being located at cyclone centerline on a plane level with the bottom of the feed nozzle. A 90° elbow joined the vertical overflow section to a horizontal outlet tube passing through the cylindrical section wall. The centerline of the horizontal section was 8 1/2 in. above that of the feed nozzle. Withdrawing the overflow above the feed nozzle preserves the potential head of the fluid, making it possible to utilize gravity flow in plant applications.

The horizontal overflow section, because of its location cross-current to the flow, caused considerable turbulence with resulting breakdown of flow pattern along the downstream side of the pipe. Classification was impaired and capacity per unit of fluid energy was lowered. An airfoil of galvanized iron was constructed following dimensions given for a standard streamline form with a fineness ratio (length to maximum diameter) of 2.⁵ The airfoil enveloped the horizontal section from junction at cyclone wall to edge of elbow. Turbulence along the trailing edge was greatly diminished, although further improvement is desirable. Data will be shown to indicate the advantages obtained by this design improvement.

The cyclone was originally constructed with a peripheral weir discharge 7 in. above the top of the overflow horizontal section. This discharge was used at high feed rates, but it was found that it contained coarser solids than the regular overflow stream. Accordingly, a 2-ft extension was added to the cylindrical section, making a weir discharge unnecessary. Evidence of the superiority of this construction will also be proved later by the experimental data. The frontispiece illustrates the open-top cyclone construction after the 2-ft cylindrical extension had been added.

All runs were made on slurries containing ground agricultural dolomite of 2.83 sp gr. All particles were uniform in shape and specific gravity, and less than 1 pct were coarser than 20 mesh. The solids were charged to a 250-gal open tank equipped with a high speed mixer and water added to obtain correct solid concentration. A slurry pump fed the pulp to the open-top cyclone through a hand-operated throttling gate valve for control of flow

rate. The overflow, underflow, and weir discharge returned to the slurry tank through launders or sheet metal conduits. Before any analyses or rate samples were taken, the system was allowed to pump for several minutes to attain equilibrium.

Samples were taken from all discharge streams to secure data on size distributions and solid concentrations. Rates were determined by collecting a discharge stream in a weighing tank for a timed interval. Solid concentration and size distribution of the feed stream were obtained by back calculation from the discharge streams.

Effect of Overflow Rate and Volume Split: After many exploratory and test runs, it was established that the sharpest and finest classifications were obtained with a streamlined horizontal overflow section, and only one overflow discharge originating at the cyclone centerline. Several runs, therefore, were made at varying feed rates and volume splits to the underflow with constant feed solid concentration. Results are indicated in Table I. Range and average size distribution for feeds of Table I are given in Table II and indicate sufficient uniformity. It will also be observed in Table I that feed solid concentration varied between 22.0 and 24.5 pct

Table II. Size Distribution of Feed Solids in Table I

Size Fraction	Range, Wt, Pct	Average Wt, Pct
+20 mesh U.S. standard	0.5 to 0.8	0.6
20x40	30.8 to 38.0	34.4
40x60	18.1 to 20.8	19.4
60x100	6.5 to 10.2	8.3
100x140	2.9 to 3.7	3.3
140x200	2.6 to 3.6	2.9
-200	29.1 to 33.8	31.1

only for all runs except run 119, which exhibited an overloaded underflow discharge and will be considered separately.

Two particle dimensions were calculated from the data: 1—diameter of separation, the diameter in microns at which 1 1/2 pct by weight of the overflow solids would be coarser; and 2—equilibrium particle, the diameter of the particle in microns which is in equilibrium with the fluid currents at the critical separation point within the open-top cyclone. Theoretically, it is the particle size whose settling velocity is exactly equal to and opposite that of the radial velocity of the fluid at the radius

separating the inner and outer spirals of the free vortex within the cyclone under the conditions of operation. It can be calculated as follows. Any water withdrawn in the underflow must carry with it at least the same percentage of feed solids (heavier than the slurry density) of each diameter as the per cent of feed water reporting to the underflow. Therefore, the recovery per cent in the underflow for the equilibrium particle will be:

$$X = \frac{100-Y}{2} + Y = 50 + \frac{Y}{2} \quad [1]$$

Where X = weight per cent recovery of the equilibrium particle size in the underflow

Y = per cent of feedwater reporting to the underflow.

Accordingly, from the underflow recovery curve for each run, the equilibrium particle size would be determined at the above per cent.

The diameter of separation and equilibrium particle size for Table I are plotted against the overflow rate in gallons per minute in Fig. 1. A small increase in equilibrium particle size is observed with an increase in flow rate, while diameter of separation shows a slightly steeper correlation. Both curves appear to be asymptotic in nature; the reverse is true for the conventional liquid-solid cyclone. Both curves would exhibit a decrease as overflow rate increased for a pressurized cyclone of fixed dimensions, in as much as centrifugal force increases faster than radial forces tending to sweep particles to the overflow.^{1,8} Because of the relatively small fluid forces involved in the open-top cyclone, probably a relatively high percentage of fluid energy is consumed in overcoming wall friction, which is negligible in the conventional cyclone. Thus a smaller percentage of fluid energy is converted into centrifugal force so that the increased radial forces at higher flow rates cannot be entirely offset. However, it may be possible by improved design, i.e., smaller included angles and better streamlining of overflow section, to reduce

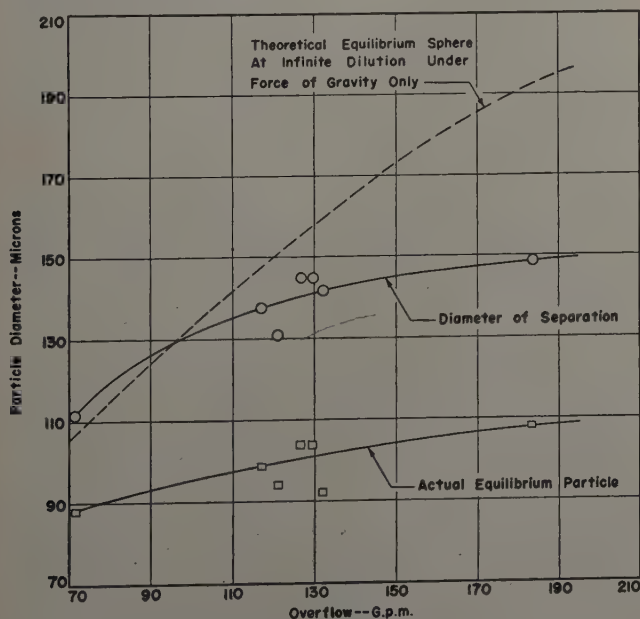


Fig. 1—Diameter of separation and equilibrium particle size as a function of overflow rate for the 30-in. experimental open-top cyclone. Feed solid concentration range, 22.0 to 24.5 wt pct.

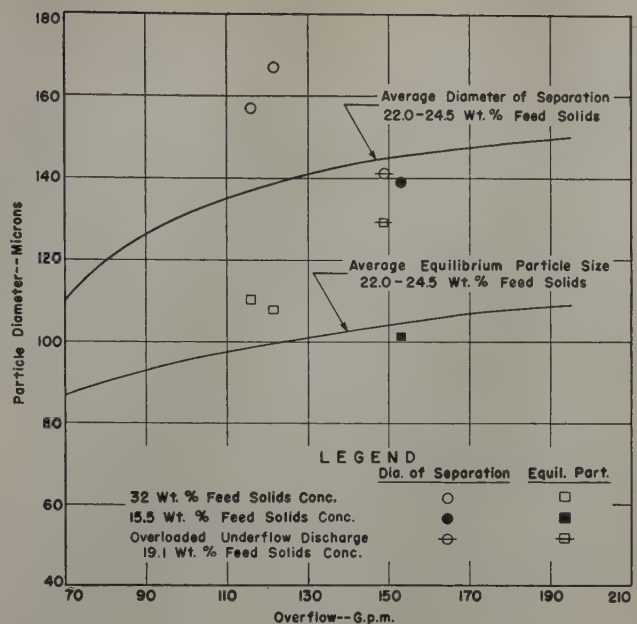


Fig. 2—Effect of feed solids concentration and type of underflow discharge on diameter of separation and equilibrium particle size for the 30-in. experimental open-top cyclone.

wall friction and obtain a diameter of separation or equilibrium size independent of overflow rate.

The theoretical equilibrium sphere curve at infinite dilution under the force of gravity only for the conventional gravitational classifier has been included in Fig. 1. This was done by calculating the diameter of a sphere of specific gravity 2.83 whose settling velocity at infinite dilution is equal to the overflow rate in cubic feet per minute divided by the cross-sectional area in square feet of the open-top cyclone. This curve would represent theoretical minimum equilibrium particle size for gravitational classifiers operating on this solid and would naturally be exceeded under hindered settling conditions.

It should be pointed out that the equilibrium particle for conventional gravitational classifiers is generally felt to be best expressed by diameter of separation. All particles coarser than this size will be given a greater settling velocity, by the constant force of gravity, than that of the opposing fluid currents. However, this is not true for a free vortex, as neither the centrifugal force nor radial velocity is constant across any normal plane to the centerline. Accordingly, equilibrium separation occurs at the division between the inner spiral, directed toward the overflow, and the outer spiral, directed toward the underflow. It is hypothesized that this is best represented for a free vortex by the equilibrium particle size calculated from Eq. 1. On the basis of equilibrium, comparison would have to be made between diameter of separation for gravitational classifiers and what has been called equilibrium particle size for the open-top cyclone. However, owing to the general recognition of the importance of diameter of separation in analyzing classification results, comparison can better be made on this basis.

It will be observed that the equilibrium curve at infinite dilution for gravitational classifiers for the most part lies well above the diameter of separation of the open-top cyclone. In a gravitational classifier there would definitely have been hindered settling with feed and overflow solid concentrations

similar to those found with the open-top cyclone. Thus improvement will be even better. Furthermore, the slopes of the two curves of Fig. 1 for the open-top cyclone are considerably less than the theoretical curve for infinite dilution under the force of gravity only. It can be safely concluded that a free vortex pattern has been established which effectively adds centrifugal force to force of gravity to obtain a finer classification point.

It will be noticed in Table I that underflow solid concentration was allowed to vary from 52.0 to 74.5 pct without any appreciable effect on its diameter of separation or equilibrium particle size. This is similar to results obtained with conventional liquid cyclones.⁸

Sharpness of classification is indicated by efficiency of separation calculated from a formula by Taggart.⁹ Efficiencies ranged from 76.9 to 84.6 pct, which is generally considered high compared with efficiencies for gravitational classifiers. However, classification is greatly affected by size distribution and feed solid concentration, so that as Taggart cautions,⁹ efficiency of classification expressed as a percentage is relatively unimportant by itself. Other data are necessary for purposes of comparison.

As a better indication of the sharpness of separation with the open-top cyclone, it will be observed in Table I that the diameter of separation is equal to or finer than 100 mesh in all cases. Simultaneously, it was found that the percentage of the -200 mesh fraction of feed recovered in the underflow was in all cases equal to or very slightly greater than the per cent of feed water recovered in the underflow. As this represents the absolute minimum recovery of any size fraction, it is apparent that the underflow recovery curve breaks from a maximum of 100 pct to an absolute minimum over a very close size range.

Effect of Feed Solids Concentration and Type of Underflow Discharge: Three runs, reported in Table III, were made at feed solids concentrations ranging from 15.5 to 32.1 pct. Diameter of separation and equilibrium particle size were calculated as indicated in Table III and are also plotted in Fig. 2 as a function of overflow rate. The average curves of Fig. 1 for diameter of separation and equilibrium particle size at 22.0 to 24.5 pct feed solids concentration are reproduced on Fig. 2 for comparison.

From Fig. 2 it is apparent that feed solids concentration of 32 pct increased diameter of separation and equilibrium particle size over those at 22.0 to 24.5 pct as would be expected. Undoubtedly this is due to more intense hindered settling conditions. Diameter of separation increased by an average of 17.4 pct, while equilibrium particle size increased

only 9.8 pct. These increases are well within values that would be predicted by hindered settling theory on gravitational classifiers.

With the above results, it would be logical to expect a lowered classification efficiency. As seen from Table III, efficiency was only 75.6 pct as compared with an average of 82 to 83 pct for the 22.0 to 24.5 pct feed solid concentration. Naturally, some of this drop in efficiency is also due to removal of a greater percentage of feed water in the underflow with the heavier solid concentration.

The results of Run 123 at only 15.5 pct feed solids concentration are slightly lower than the average conditions of 22.0 to 24.5 pct feed solids concentration. However, as only one run was made, this could be a statistical deviation. From previous conventional cyclone work, increase in equilibrium particle size on this material would be expected to begin at about 20 pct feed solids concentration.

During the exploratory runs overloaded underflow discharges were obtained in a few cases, but it was observed that overflow solids did not appear to increase in top size. This is contrary to experience with conventional cyclones with overloaded discharges wherein solid recovery in the underflow is very severely decreased and most of the particles are lost to the overflow. Run 119 of Table I was performed with an overloaded underflow discharge at 19.1 pct feed solids concentration to secure quantitative information. It will be observed from Fig. 2 that equilibrium particle size increased about 25 pct over that of the average results at 22.0 to 24.5 pct feed solids concentration and a vortex underflow. However, diameter of separation was slightly below the average, indicating that coarser particles were not being lost to the overflow. From these results it is also apparent that classification occurred over a relatively narrow size range, with the result that classification efficiency at 100 mesh was a very high 91.5 pct.

The explanation for the continued high recovery above the vortex underflow mesh of separation, even with overloaded discharges, probably lies in the much lower centrifugal forces utilized in the open-top cyclone. Thus when an overloaded underflow occurs, centrifugal force is not sufficient to *twist-off* the inner spiral at the conical section apex to severely impair solids discharge. A similar phenomenon has been experienced with large conventional cyclones, 36-in. diam, operating at only 3 lb per sq in. gage feed pressure.

It is not argued that an overloaded discharge will never increase the mesh of separation of an open-top cyclone. Naturally it would be possible to feed solids at such a rate that they could not be withdrawn fast enough through an undersized underflow. Rather, it indicates that the open-top cyclone should be able to handle a wider variation in solids rate than the conventional cyclone without impairing mesh of separation. Furthermore, it was observed that there was never plugging of the underflow spigot, as can occur with some gravitational classifiers.

Effects of Non-Streamlined Overflows: Several runs were made initially without the streamlined horizontal overflow section. Data for Runs 104 and 106 are given in Table IV with operating conditions comparable to those of Table I and Fig. 1 using the streamlined overflow. Diameter of separation averaged 17 pct higher and equilibrium particle size 9.5 pct higher than the corresponding values for

Table III. Open-Top Cyclone, Effect of Feed Solids Concentration on Classification, All Solids 2.83 Sp Gr Dolomite

	Run 121	Run 122	Run 123
Slurry Rate, GPM			
Feed	159.1	157.6	164.0
Overflow	121.2	116.0	153.1
Underflow	37.9	41.6	10.9
Solids Concentration			
Feed	31.35	32.1	15.5
Overflow	10.98	11.72	8.95
Underflow	69.6	67.2	69.6
Solids Rate, Tons Per Hr			
Feed	15.50	16.19	7.11
Overflow	3.55	3.74	3.66
Underflow	11.95	12.45	3.45
Diameter of Separation, Microns	167	157	139
Equilibrium Particle, Microns	108	110	102
Classification Efficiency, Pct at 100 Mesh	75.8	75.4	86.4
Liquid Level Above Bottom of Horizontal Overflow Section, In.	17	14	21½

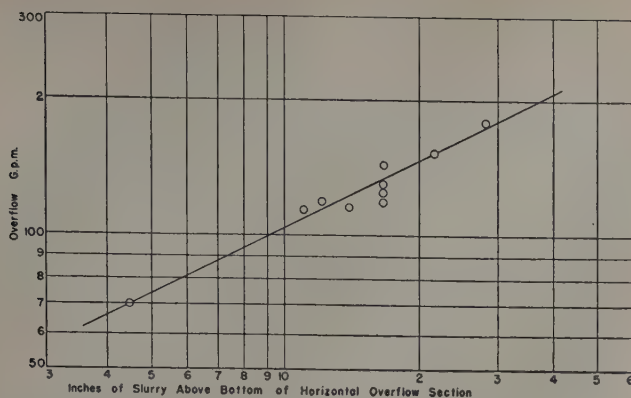


Fig. 3—Energy requirements for open-top cyclone, experimental 30-in. unit.

the streamlined overflow. As would be anticipated, therefore, classification efficiency was also lowered.

The flow pattern was obviously more turbulent with the non-streamlined overflow, indicating that more of the initial fluid energy was not available for the centrifugal force. Accordingly the above results are to be expected. Consequently every effort should be made to present as little obstruction to the free vortex flow pattern with the open-top cyclone. This necessitates not only streamlined overflows but also smooth wall construction and continuous taper of the conical section.

Effect of Weir Overflow Discharges: Several runs were also made initially with a peripheral overflow weir discharge as well as a simultaneous central non-streamlined overflow discharge. Table V shows results for Runs 107 and 112, whose operating conditions are similar to runs of Table I and Fig. 1 with only a central streamlined overflow discharge. Equilibrium particle size averaged 16.2 pct higher than with the streamlined central overflow. However, diameter of separation was considerably above that of the streamlined central overflow. Weir discharge averaged 99.7 pct higher, the simultaneous central overflow averaged 43.4 pct higher, and the composite averaged 57.8 pct higher. While the non-streamlined overflow contributed to a portion of this injury, it is obvious from the previous section that greatest impairment arose from the weir discharge.

The peripheral weir receives fluid from the outer radius of the free vortex. This radius possesses the lowest centrifugal force of the outer spiral wherein all separation must be achieved. Consequently, the higher diameter of separation is not surprising. However, the free vortex pattern is also distorted by such a weir discharge. A high percentage of the energy of the fluid so discharged is not available for the creation of centrifugal force within the unit. Consequently, the central overflow discharge also suffers severely. This is easily proved by the higher diameter of separation in the central overflow of Runs 107 and 112 when compared with results of Fig. 1. Accordingly finest separations can be expected in the open-top cyclone when the optimum creation of a free vortex is followed. A single central overflow from the inner spiral of the vortex is therefore desirable.

Energy Requirements: Because of the very low feed pressures involved, below 1.5 lb per sq in., it was impossible to obtain reliable data on feed pressures to the open-top cyclone. Height of the outer liquid surface, which is proportional to energy

Table IV. Open-Top Cyclone, Sample Results without Streamlined Overflow Section and Comparison with Streamlined Overflow Results

	Run 104	Run 106
Slurry Rate, GPM		
Feed	116.7	125.2
Overflow	94.9	102.8
Underflow	21.8	22.4
Solid Concentration, Wt, Pct		
Feed	17.5	20.7
Overflow	6.83	7.01
Underflow	50.2	60.7
Solids Rate, Tons Per Hr		
Feed	5.73	7.47
Overflow	1.68	1.87
Underflow	4.05	5.60
Diameter of Separation, Microns	149	146
Diameter of Separation with Streamline Overflow, Microns	129	132
Equilibrium Particle Size, Microns	110	98
Equilibrium Particle Size with Streamline Overflow, Microns	94	96
Classification Efficiency at 100 Mesh, Pct	71.0	68.3
Estimated Classification Efficiency at 100 Mesh with Streamlined Overflow and Similar Underflow Solid Concentration, Pct	76.0	80.0

requirements, was accordingly noted with the single streamlined overflow runs, see Tables I and III.

The best correlation for energy requirement was obtained by plotting the log of overflow rate in gallons per minute against the log of the height of slurry above the bottom of the horizontal overflow section as indicated in Fig. 4. A line of slope 0.5, which would be anticipated by fluid mechanics theory, fit the data with sufficient accuracy. The equation for this straight line is:

$$\text{gpm} = 115\sqrt{F} \quad [2]$$

where gpm = overflow rate, gal per min
 F = height of slurry above bottom of horizontal overflow section, in feet of overflow slurry.

In a previous paper, the expression $\text{gpm}/\sqrt{F}$ has been termed the capacity ratio. For a conventional cyclone, it was found equal to the following:

$$\frac{\text{gpm}}{\sqrt{F}} = K (eb)^{0.0} \quad [3]$$

where e = overflow diam, in.

b = inlet diam, in.

K = capacity ratio proportionality constant.

Table V. Open-Top Cyclone, Sample Results with Weir Discharge and without Streamlined Overflow Section. Comparison with Streamlined Overflow and without Weir Discharge at Same Capacity

	Run 107	Run 112
Slurry Rate, GPM		
Feed	200.0	212
Overflow	133.3	128
Weir	47.0	63.3
Underflow	19.7	20.7
Solid Concentration, Wt, Pct		
Feed	16.05	18.3
Overflow	7.71	9.9
Weir	6.93	9.2
Underflow	64.5	67.1
Solid Rate, Tons Per Hr		
Feed	8.93	11.09
Overflow	2.70	3.38
Weir	0.85	1.54
Underflow	5.38	6.17
Diameter of Separation, Microns		
Overflow	197	232
Weir	295	302
Composite	222	250
Without weir discharge and with streamlined overflow at same capacity	149	150
Equilibrium Particle Size, Microns	118	133
Equilibrium Particle Size, Without Weir Discharge and with Streamlined Overflow at Same Capacity, Microns	108	109
Classification Efficiency at 100 Mesh, Pct	80.5	81.6
Estimated Classification Efficiency at 100 Mesh without Weir Discharge and with Streamlined Overflow at Same Capacity, Pct	84.0	84.0

Using this equation with appropriate diameters of feed and overflow for the open-top cyclone, the capacity ratio proportionality constant would be 9.38. Conventional cyclones will normally exhibit values of this constant from 5.5 to 6.5. Accordingly, on the basis of the above equation the experimental open-top cyclone permitted a 44 to 71 pct higher capacity per unit of energy input. However, it is felt that the last equation will not hold for the open-top cyclone. Instead, either the 0.9 exponent will be changed to a higher value or the capacity ratio will be a function only of the overflow diameter. Further experimental work will be needed to verify this. At any rate, the higher capacity ratio is to be expected from previous comments. Wall friction percentage is felt to be considerably more important with the open-top cyclone than with conventional cyclones. As it is easier to overcome wall friction than to generate a free vortex, the higher throughput per unit of input energy for the open-top cyclone is entirely logical.

It should be pointed out that the greatest submergence used in these runs was 28 in. above the bottom of the horizontal overflow section. To obtain feed pressure at the feed entrance to the open-top cyclone, it is necessary to add the distance from the centerline of the inlet to the bottom of the horizontal section of the overflow. The experimental unit had a separation of only 6½ in., making the maximum feed pressure less than 3 ft of overflow slurry. This compares favorably with any gravitational classifier. Furthermore, as the overflow is discharged near the top of the cyclone, much of the original feed energy is still available in the form of potential head.

Comparison with Other Gravitational Classifiers: Data from the section of classification in the *Handbook of Mineral Dressing* by Taggart was used to compare the results of the open-top cyclone with other gravitational classifiers.⁷ Summaries of performances of four different classifiers will be found in Table VI together with representative runs on the open-top cyclone. Diameter of separation, classification efficiency, overflow gallons per minute per square foot of settling pool area, and tons of feed solids per hour per square foot of settling pool

area were also calculated and listed in Table VI. It should be emphasized that performance runs were selected so that operating conditions were as closely comparable as possible with respect to solid specific gravity, feed solids concentration, and diameter of separation. Because of this specification, it was not possible to obtain comparisons for all types of gravitational classifiers.

It is not intended to imply that the open-top cyclone is entirely superior to other gravitational classifiers. Naturally each application will designate the type of classifier to be used. Rather, the comparisons will be made with respect to what is believed to be the specific advantage of the open-top cyclone: higher capacity per square foot at a relatively stable mesh of separation, with possibly sharper classifications.

All performance tests but one exhibited a mesh of separation of 100 or 150 mesh, comparable to results with the open-top cyclone. Rake classifiers averaged 2.9 gpm of overflow per sq ft of settling pool area with a high of 4.6. Bowl-rake classifiers averaged 4.2 gpm per sq ft with a high of 7.35. The one test on the Allen cone gave 1.8 gpm per sq ft while the Callow tank yielded 3.1 gpm per sq ft. The open-top cyclone ranged from 14.5 to 37.3 gpm per sq ft with an average of 27.4. This average is 552 pct higher than the highest average of the Bowl-Rake classifiers. As has already been pointed out, moreover, the open-top cyclone can accept a severe increase in feed rate with only a small change in diameter of separation. Thus the 37.3 gpm per sq ft is a realistic figure which is 789 pct above the highest gravitational classifier average. If higher slurry levels above the overflow pipe can be tolerated, still higher capacities per square foot can be obtained.

Rake classifiers averaged 0.47 tons of feed solids per hr per sq ft of settling pool area with a high of 0.55, bowl rake classifiers 0.29 tons with a high of 0.55, the Allen cone 0.28 tons, and the Callow tank 0.18 tons. The open-top cyclone averaged 2.11 tons per hr per sq ft for the 5 representative runs, with a high of 3.30. This average is 349 pct above the highest exhibited by rake classifiers. Because of the classification stability of the open-top cyclone with

Table. VI. Comparison of Operating Data for Industrial Classifiers with Open-Top Cyclones

Type of Solid	Solid Sp Gr	Solids, Wt. Pct			Feed	GPM		Solids, Tons Per Hr			Diameter of Separa- tion, Microns	Classification Pool		Over- flow, GPM Per Sq Ft	Solids, Tons Feed Per Sq Ft Per Hr	
		Feed	Over- flow	Under- flow		Over- flow	Under- flow	Feed	Over- flow	Under- flow		Efficiency, Pct*	Area, Sq Ft²			
Rake Classifier—Taggart, Table 6, Section 8																
Siliceous	2.7	27.6	20	82	433	402	31	36.3	23.0	13.3	149		87.3	4.6	0.42	
Quartz		37.2	23	74	338	272	66	41.2	18.3	22.9	173	40.5(100)	87.3	3.1	0.47	
Siliceous	2.6	45.8	20	73	241	146	95	38.3	8.3	30.0	237	46.5(65)	91.2	1.6	0.42	
Quartz	2.7	46.7	12	82	290	200	91	50.2	6.5	43.7	142	47.7(100)	91.2	2.2	0.55	
Bowl-Rake Classifier—Taggart, Table 18, Section 8																
Siliceous		18	10	74	828	780	48	37.5	20.8	16.7	149	67.2(100)	106	7.35	0.35	
Siliceous	2.6		10			641		83.8	17.1	66.7	144	38.0(100)	153	4.2	0.55	
Siliceous	2.7	21	5	70	665	645	20	14.6	8.35	6.25	102	70.9(150)	491	1.3	0.03	
Siliceous		17	9	72	856	789	67	41.7	18.8	22.9	146	71.3(100)	201	3.9	0.21	
Allen Cone—Taggart, Table 26, Section 8																
		35	14	70.5	70.5	51.8	18.7	7.91	2.00	5.91	151	40.9(100)	28.3	1.8	0.28	
Callow Tank—Taggart, Table 30, Section 8																
		18.4	13.7	54	169	156	12.9	8.87	5.83	3.04	108	78.3(150)	50.03	3.1	0.18	
Open-Top Cyclone																
Dolomite	2.83	22.6	9.00	74.5	209.3	183.2	26.1	13.87	4.37	9.50	149	84.1(100)	4.91	37.3	2.82	
Dolomite	2.83	23.5	9.20	66.0	85.5	71.0	14.5	5.91	1.73	4.18	112	82.9(140)	4.91	14.5	1.20	
Dolomite**	2.83	19.1	9.74	75.7	162.3	148.8	13.5	8.81	3.85	4.96	138	91.5(100)	4.91	30.2	1.80	
Dolomite	2.83	32.1	11.72	67.2	157.6	116.0	41.6	16.19	3.74	12.45	154	75.4(100)	4.91	23.6	3.30	
Dolomite	2.83	15.5	8.95	69.6	164.0	153.1	10.9	7.11	3.66	3.45	135	86.4(100)	4.91	31.2	1.45	

* Numbers in parentheses indicate the mesh at which efficiency is calculated.

** Run operating with an overloaded underflow discharge.

respect to variations of feed rate, higher solids rates than the above average are entirely feasible.

It is believed that open-top cyclone capacity per square foot of area can be increased still further. Pressurized cyclone performance has indicated that inlet and overflow nozzles should cover approximately 60 pct of the cyclone diameter to obtain minimum classification points. In other words,

$$\frac{e + 2b}{D} = 0.6$$

where D = cyclone diameter.

In the experimental open-top cyclone with 4-in. inlet and overflow nozzles, only 40 pct of the 30-in. diam was consumed. Accordingly, it is felt that a 20-in. diam could have been employed in this work. This would decrease cross-sectional area to only 44.5 pct of the experimental unit while increasing overflow and solids rates per square foot to 2.25 times those actually experienced. Overflow rate of 84 gpm per sq ft may be entirely realistic. This is 1900 pct above the highest average of the gravitational classifiers.

With the previously mentioned warning that efficiency of classification can be misleading, it will be noticed that only one gravitational classifier test exhibited a percentage as high as the open-top cyclone. Classification efficiencies at mesh of separation for gravitational units ranged from 38.0 to 78.3 pct, while the open-top cyclone ranged from 75.4 to 91.5 pct. From this comparison a sharper classification with the open-top cyclone appears possible.

No comparison was made with respect to power requirements, but it is believed that the maximum submergence of the feed pipe of 34½ in. of slurry used with the open-top cyclone is reasonably low. Furthermore, as no moving parts are involved, no power is required for rakes or motor-operated valves.

Summary

A different approach to gravitational classification equipment has been proposed to afford more *self-regulation* to the operation, enlarge capacity per square foot, and possibly increase sharpness of classification at no increase in power requirements. Experimental evidence indicates that this can be achieved by creating a free-vortex through the use of a tangential feed entry to a vessel somewhat similar to a conventional cyclone. However, no cover is used and vessel diameter will greatly exceed that of conventional cyclones. Results indicate also that the theory of this open-top cyclone will differ in many respects from that of the smaller pressurized cyclones. Some of this theory has been expressed from the experimental results obtained to date, but further work is necessary to a more complete fundamental understanding. The following conclusions are indicated at this time.

1—The open-top cyclone exhibits a mesh of separation and an equilibrium particle size well below the expected value for conventional gravitational classifiers on the basis of overflow capacity per square foot of pool area. Mesh of separation on 2.83 sp gr solids in these tests ranged from 100 to 140.

2—Mesh of separation and equilibrium particle size are considerably more immune to feed rate fluctuations than gravitational classifiers.

3—A peripheral weir discharge should not be utilized with an open-top cyclone if a minimum

classification point is desired. Such a discharge arises from the lowest centrifugal force radius of the free vortex and simultaneously robs fluid energy from the vortex pattern.

4—A single, centrally originating overflow is desirable with a streamlined horizontal section to eliminate as much turbulence as possible. Furthermore, the open-top cyclone design should minimize all wall friction by maintaining smooth constant tapers in the conical section.

5—Slurry level above the feed pipe can be held to less than 3 ft, making energy requirements relatively unimportant. Possible energy equations have been proposed but remain to be obtained by further experimental data on different size units.

6—Overflow rates expressed as gallons per minute per square foot of settling pool as high as 37.3 have been obtained. This is 789 pct above the highest average of gravitational classifiers operating under similar conditions. It is believed that rates of 84 gpm per sq ft are possible by covering a larger percentage of cyclone diameter with inlet and overflow nozzles than were employed in the experimental unit.

7—Solid feed rates averaged 2.11 tons per hr per sq ft of settling area with a high of 3.36. This average is 349 pct above highest average for gravitational classifiers. These also could be increased 2.25 times, for the above reason.

8—Evidence points to higher classification efficiency with the open-top cyclone.

9—No plugging of the underflow spigot occurred and the open-top cyclone was operated with an overloaded discharge without injuring the mesh of separation.

10—Abrasion has been a difficult problem with conventional cyclones in certain installations. However, the open-top cyclone operates with much lower solid velocity at the underflow, where most of the conventional cyclone wear has occurred, owing to the low fluid forces involved. It is expected that abrasion problems will accordingly be decreased.

Acknowledgments

This paper would not be complete without the acknowledgment of the helpful advice and encouragement obtained from Mr. R. L. Sutherland of Truax-Traer Coal Co. Thanks are also due Mr. Richard Nodorp, a chemical engineering student at Northwestern University, for valuable assistance in performing many of the test runs.

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Crushed Limestone Aggregates for Concrete

by Katharine Mather

This paper is an attempt to put together petrographic, physical, and chemical data about the large and varied group of rocks generally called limestones. Results of the properties of these rocks on their performance as concrete aggregates are discussed.

CRUSHED stone classified as limestone comprises about 70 pct of the crushed concrete aggregate and roadstone produced in this country, and the product has amounted in the recent past to about 97 million tons per year, valued at more than \$123,000,000.¹ Therefore an evaluation of the properties of rocks classified in these statistics as limestones and an ability to predict their behavior as aggregate in portland cement concrete are matters of considerable economic consequence to producers and consumers of aggregate. This paper is an attempt to put together petrographic, physical, and chemical data about the large and varied group of rocks generally called limestones. It also points out certain results of the properties of these rocks on their performance as concrete aggregates.

The problems encountered in efforts to explain the behavior of limestones, dolomites, and related rocks as concrete aggregate lie on the frontiers between the ignorance of the geologists and the ignorance of the engineers. The mineral composition and textures of the rocks have not been adequately explored and described, and the effects of mineral composition and texture on the performance of aggregates have been worked out in only a few cases. There are instances of unsatisfactory performance and some generalizations of those experiences. There are many more examples of satisfactory performance than of unsatisfactory, but there is practically no published information as to why a particular aggregate was satisfactory in a specific use. There are all too few investigations of undeteriorated concrete; more is known about the pathology of concrete than about its normal structure and aspect.

As part of the investigation of aggregates, the classification of rocks is not an end in itself, but is a step toward recognition of properties expected to influence the behavior of aggregates in their intended use. Rock of any type may perform well or poorly as concrete aggregate, depending on the physical condition of the rock, its physical and chemical properties as compared to those of the matrix in which it is placed, and the circumstances under which the concrete is exposed. Some classification and descrip-

tion of the limestones is necessary to distinguish subgroups whose members will probably behave in similar ways and to evaluate the group as a whole.

The group name, limestone, includes and tends to conceal the identity of rocks that have a very great range in chemical and mineralogical composition, structure and texture, conditions of deposition, subsequent history, and suitability for use as concrete aggregate. All the rocks classified as limestones include one or both of the most abundant rock-forming carbonate minerals, calcite and dolomite, as major constituents. *Carbonate rocks* is a more descriptive and more accurate group name than limestone, which has generally been used to include dolomitic or magnesian limestones and dolomites as well as calcitic limestones.

Classification and Description of the Carbonate Rocks

The question of names to be applied to the carbonate rocks may appear academic, but it is of practical importance as long as there are specifications for aggregate that result in acceptance or rejection of rocks by name and as long as failure to make distinctions among the carbonate rocks leads to misunderstandings of their properties and of what may be expected of them in performance. Materials included in production statistics as limestone and tested for concrete aggregate as limestone range in fact from calcareous shale and calcareous or dolomitic sandstone and argillaceous limestone and dolomite to calcitic limestone and pure dolomite and to rocks close to calcitic and dolomitic marble. To take a simple example: consider two manufactured coarse aggregates, both dense and unweathered, one composed principally of crystalline calcite and the other of crystalline dolomite, but both known as limestone. If each is used as coarse aggregate in a standardized concrete mixture with the same fine aggregate used in both cases, made into concrete specimens, and tested in accelerated freezing-and-thawing, the test results will probably be quite different, because of the differences in internal bond and thermal properties between the two aggregates. As long as both aggregates are called limestone, the test results cannot be explained, or used except in a very limited way; if the aggregates are distinguished as limestone and dolomite and the consequent differences in texture and thermal properties are recognized, the results are reasonable.

Classification of the carbonate rocks used as aggregate should be based on structure and on composi-

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Discussion on this paper, TP 3616H, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, January 27, 1953. Los Angeles Meeting, February 1953.

tion and texture. These last two are fundamental properties² which can be related to aggregate performance. The relation of mineral composition to performance is quite often clear and easily recognizable. Less attention has been paid to texture, probably because it is hard to describe, almost infinitely varied in the carbonate rocks, and often difficult to explain. Structure, particularly bedding, cleavage, and jointing should be considered as far as the samples or exposures permit because of its influence on particle shape and sometimes on the maximum size that can be produced.

The important groups of constituents to be taken into account are the carbonate minerals, the detrital constituents other than clay, the clay minerals, and the remaining, largely secondary constituents. Ordinarily either calcite or dolomite or both are the carbonates present, but others, for example siderite, are possible. It may be important to know the identity of the carbonate or to recognize the presence of both calcite and dolomite and the way they are distributed, since there are differences in thermal properties between them and differences in texture and interlock between calcitic limestones, dolomitic limestones, and dolomite. These in turn cause differences in pore structure and internal bond. The most probable detrital constituents are quartz, chert, and feldspar; various ferromagnesian minerals and opal have been found. Some of the clay minerals can have very marked effects on aggregate performance even when they are present in very small amounts, depending on their type, distribution, and accessibility to water. The remaining constituents ordinarily found are secondary chert or quartz; sulphides (pyrite and marcasite), sulphates (anhydrite, gypsum), phosphates (carbonate apatites), and hydrocarbons are among the possibilities that have been encountered.

In practice these four generalized groups may often be simplified; many carbonate aggregates can be described in terms of variations among four end members, calcite, dolomite, sand or silt, and clay, or among three of those four. Fig. 1 shows a tetrahedral diagram relating the four components, as well as subdivisions of the four faces and rock names regarded as appropriate. Compositions may be read along the edges of each triangle. The diagrams are based primarily on discussions by Pettijohn.³

Description of texture should include measures of grain-size if the rock is even-grained or information about the size-ranges of the groups of constituents (the larger clastic grains, the matrix, and the cement), and about the arrangement of each group with respect to the others. Most of the textural terms current about sedimentary rocks are inadequately defined and unfamiliar to most geologists and most engineers. Their use is likely to cause confusion and annoyance. Krynine⁵ has provided a very useful discussion on how to describe texture. Sander's examples⁴ are useful and interesting but written in highly specialized language.

No new names or new classification for carbonate rocks are proposed here. The writer's experiences in trying to fit aggregate samples into classifications have led her to believe that none of the detailed classifications with which she is familiar are satisfactory for this use. The basic standardized descriptive scheme used for igneous rocks can be applied to the limestones and dolomites, and the names constructed will consist of the following parts: color, textural modifier, varietal minerals and cement, and

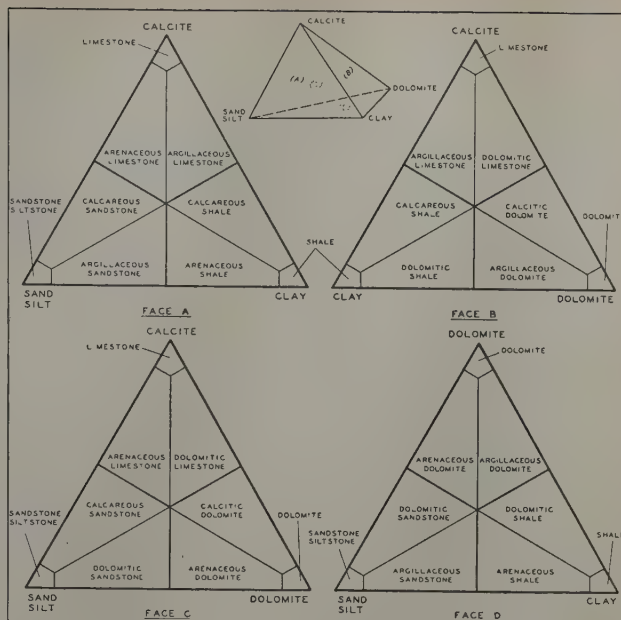


Fig. 1—Tetrahedral diagram showing variations in composition among rocks composed of calcite, dolomite, clay, and sand-silt.

principal name.⁶ All four of the generalized groups of constituents, the carbonates, detrital constituents other than clay, the clay minerals, and the secondary products, are important in assessing the potentialities of a carbonate aggregate. The texture is the second important consideration, including the ranges in grain size or crystal size of each group of constituents and their arrangement with respect to each other. The third is structure, particularly bedding, cleavage, jointing, and alternations of different lithological types. Attention to these three considerations can result in rock names that are clear, adequately defined, and objectively based. These descriptions will contain information permitting as good predictions about performance as can now be made.

Properties of Carbonate Rocks as Concrete Aggregate

It is convenient to discuss the effects of rock properties under two headings: those that influence production and those that influence use in concrete. The same properties are involved, but some of those influencing production act at that stage to eliminate potential sources, and some that make minor or negligible differences in production have marked effects in concrete.

Properties Influencing Production: To be produced as concrete aggregate, a carbonate rock must be available in adequate quantity for the intended use, must be present in beds thick enough to permit aggregate of the desired maximum size to be made, and must be strong and cohesive enough to be processed and emerge as a graded product in the size-ranges desired. The second requirement eliminates some thin-bedded deposits as sources of aggregate for mass concrete, although they are suitable for pavements or buildings. The third eliminates many of the consolidated but soft Cretaceous and Tertiary limestones of the Gulf Coast that crumble when they are crushed.

Interbedded shales in carbonate rocks give rise to problems of several kinds in both production and use. One such problem was solved by the information obtained from petrographic investigations

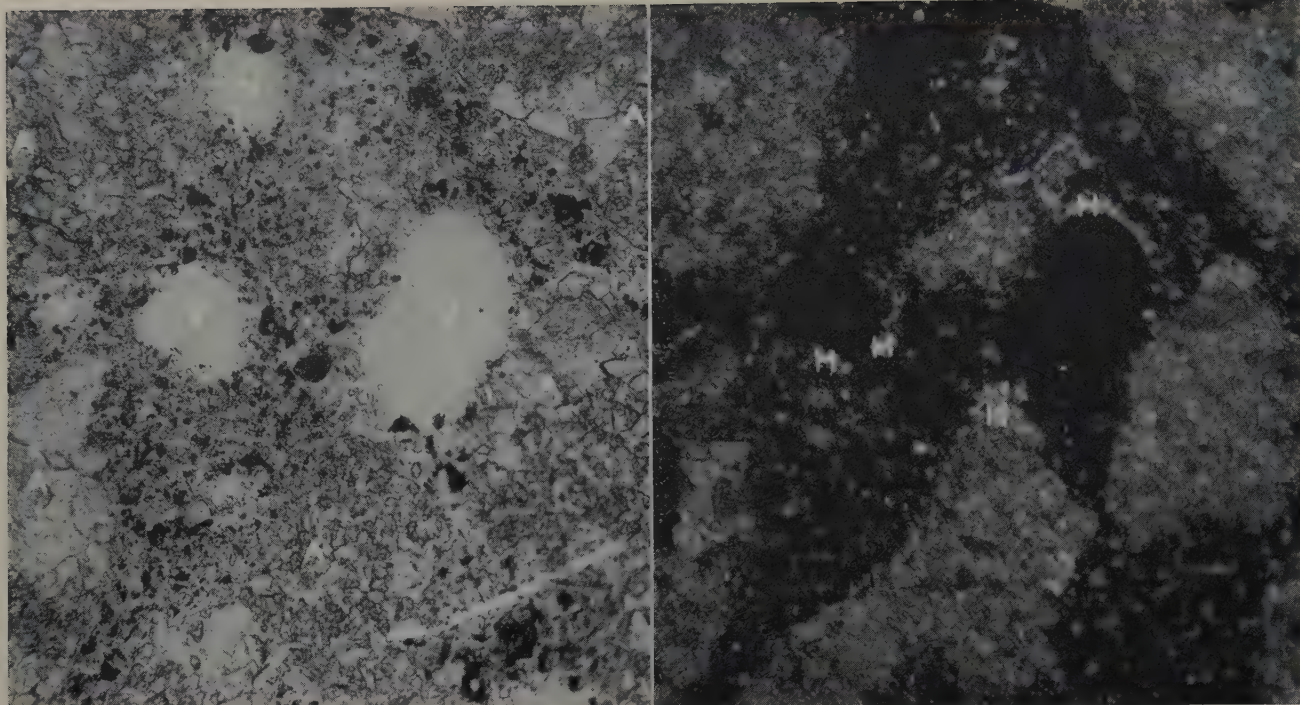


Fig. 2—Dam B, interior mass concrete containing limestone aggregate, X165, plane light (left), and crossed nicols (right). At left are shown voids (V), aggregate (A), and scattered dark residual cement grains. At right, calcium hydroxide crystals (H) form fairly small areas. The multitude of minute light dots in the groundmass are limestone dust smaller than 15 microns.

and laboratory studies of processing of aggregate for a dam in southern Kansas located about 40 miles from the nearest commercial quarry. A 12-ft section of limestone was exposed near the dam site. Blocks of ledge rock from each bed were shipped to the Waterways Experiment Station, with enough information to permit complete reconstruction of the section. The limestone was medium-grained and somewhat clayey, with open bedding planes at 4-in. to 4-ft intervals, and the rock above and below each open plane was porous and stained for thicknesses of $\frac{1}{4}$ to 1 in. Stylolites were spaced at 2-in. to 1-ft intervals; the rock broke easily on some of the stylolites but on others the bond was strong. Some of the stylolites contained soft fragile black shale which broke away easily from the limestone and disintegrated in water. Shale and porous weathered rock amounted to 10 pct of the total thickness. It appeared that much of the shale and much of the weathered porous limestone could be eliminated by crushing and washing, leaving uncoated limestone of satisfactory quality. Aggregate processed in the laboratory, made into concrete, and tested in accelerated freezing and thawing confirmed these indications. The aggregate plant installed at the site included scrubbers, and satisfactory aggregate was prepared from an apparently unsatisfactory source.

Properties Influencing Use: Performance of any concrete aggregate depends on its physical condition, its physical and chemical properties compared to those of the matrix in which it is placed, and the circumstances under which the concrete is exposed. Great variations may therefore be expected in performance of any one coarse aggregate if it is used in combination with several fine aggregates, in different structures, different concrete mixtures, and different exposures. These variations in performance are possible even if the aggregate in question is as uniform as a quarried stone can be; if the aggregate varies considerably in mineral composi-

tion and texture, as is often the case, the situation is more complicated. Considerations such as these render meaningless the attempts to classify aggregates as *good* and *bad* or *satisfactory* and *unsatisfactory* except with regard to a specific use or specific set of requirements. The same properties that make an aggregate highly serviceable in building construction or pavement may make it less desirable in mass concrete, and vice versa.

One of the properties of carbonate aggregates that varies with variation in mineral composition is linear coefficient of thermal expansion. The average linear coefficient of thermal expansion of the mineral, calcite, is 2.9×10^{-6} in. per in. per degree F. The average for dolomite is 4.2×10^{-6} . Relatively pure calcitic limestones usually have thermal coefficients in the range of 2 to 3×10^{-6} ; relatively pure dolomites have coefficients between 4 and 5×10^{-6} .⁵⁻⁷ Dolomitic limestones have coefficients of intermediate value. The average value for quartz is about 6×10^{-6} and that for chert a little lower, so that the presence of either one in appreciable quantities will raise the average thermal coefficient of either a limestone or a dolomite. A few failures of concrete ascribed to coarse aggregate of very low thermal coefficient or to anisotropic thermal expansion of coarse aggregate have been reported.^{8,9}

Callan⁶ has reported on the relations found between durability of concretes as measured by accelerated freezing and thawing of concrete beams, difference in average thermal coefficient of coarse aggregate and mortar (Δc), and absorption of coarse aggregate. Seventy-eight aggregate combinations were studied. There were 22 coarse aggregates of which 19 were calcitic limestones, and 29 fine aggregates, 18 of them manufactured limestone sands. Most of the rest were natural sands high in quartz. Each coarse aggregate had been tested in concrete in combination with several different fine aggregates. The concrete combinations made from selected

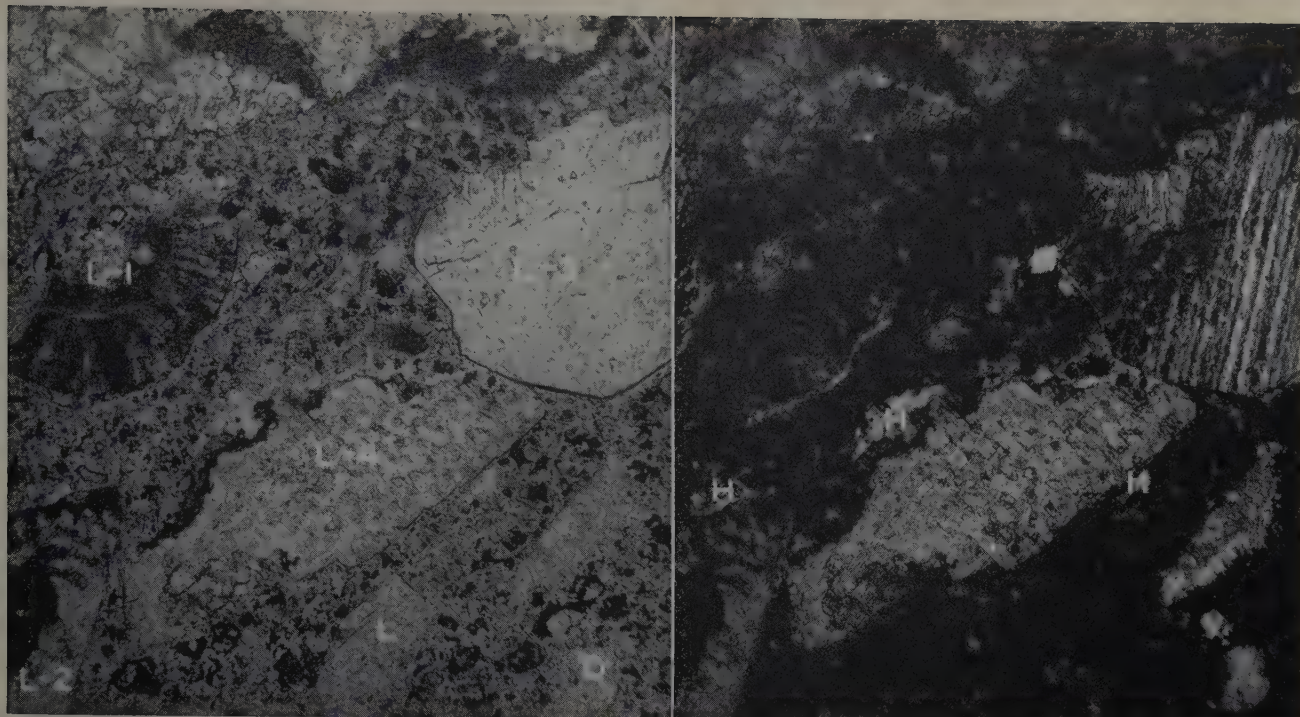


Fig. 3—Dam E, interior mass concrete, X165, plane light (left) and crossed nicols (right). At left, lower limestone fine aggregate (L) shows rhombohedral cleavage; L-3 shows twinning in plane light and with crossed nicols; quartz (Q). At right, calcium hydroxide is marked H; it also forms narrow pale gray borders along the lower edges of grains L-1 and L-3. The scattered limestone dust is smaller than in Fig. 2.

aggregates were highly standardized. All were made from Type 2 moderate-heat cement. All had a constant water-cement ratio of 5.5 gal per bag of cement, an air content of 4.5 ± 0.5 pct, and both coarse and fine aggregate of specified grading, with 1-in. maximum size stone and a ratio of 36 to 42 pct sand to total aggregate by volume. The cement content was adjusted to give the desired consistency but normally fell between 5 and 6 bags per cu yd. From each concrete mixture, three $3\frac{1}{2} \times 4\frac{1}{2} \times 16$ -in. beams were molded, cured for 9 days at 70°F and 100 pct relative humidity, and tested in an automatic freezing and thawing apparatus¹⁰ for 300 cycles, or until the average relative dynamic modulus reached 50 pct. The average thermal coefficient of each coarse aggregate and of each mortar was determined; standard physical tests were made on all the coarse and fine aggregates. Statistical analyses of the test results indicated that in this series of highly controlled concretes tested in standardized conditions, there was a multiple correlation between durability (as measured by the freezing and thawing test), difference in thermal coefficient between coarse aggregate and mortar, and absorption of the coarse aggregate. The coefficient of multiple correlation for this relation was 0.72; the coefficient of partial correlation between durability factor and difference in thermal coefficient was 0.60, and that between durability factor and absorption of coarse aggregate was 0.47. Differences in thermal coefficients between coarse aggregate and mortar were thus more important in these tests than differences in absorption.

Another investigation made at the Waterways Experiment Station provides some information on the relation of type of clay in limestone to the durability of concrete as measured by accelerated freezing and thawing tests.¹¹ This series of tests included nine carbonate coarse aggregates, seven of them calcitic limestones, one dolomite, and one dolomitic lime-

stone that contained considerable sandy limestone and calcareous sandstone. Standard physical tests and thorough petrographic examinations were made of these aggregates. Each was combined with manufactured sand from the same source, made into concrete, and tested in accelerated freezing and thawing. The coefficients of thermal expansion of each coarse aggregate and each mortar were determined. The petrographic examinations included determinations of the percentage insoluble in hydrochloric acid in each lithologic type in each sample and of the mineral composition of the insoluble residues, and micrometric determinations of the percentage of cracks and voids in each major lithologic type in each sample.

The relation between durability factor and difference in thermal coefficient between coarse aggregate and mortar that had been found in the investigation reported by Callan was also apparent here, but other relations became clear when freezing and thawing results, thermal coefficients, petrographic data, and absorption were all considered. The clay minerals found in five of the nine aggregates belonged to the kaolin group and amounted to 1 to 5 pct of the total. The other four contained montmorillonoid clays, capable of swelling; in one, the percentage was negligible; in two, there was 4 pct each, and in the last, there was 8 pct. Only one of the aggregate combinations in which the clay was kaolin had a durability factor below 80, and that aggregate had fairly high absorption and contained 15 pct cracks and voids. The aggregate that contained a trace of montmorillonoid clay had low absorption and gave a high durability factor. The 4 pct of swelling clay in each of two samples was concentrated in one porous rock type in each sample. Both gave durability factors of 75 or less. The fourth was a dolomite with 8 pct of swelling clay disseminated

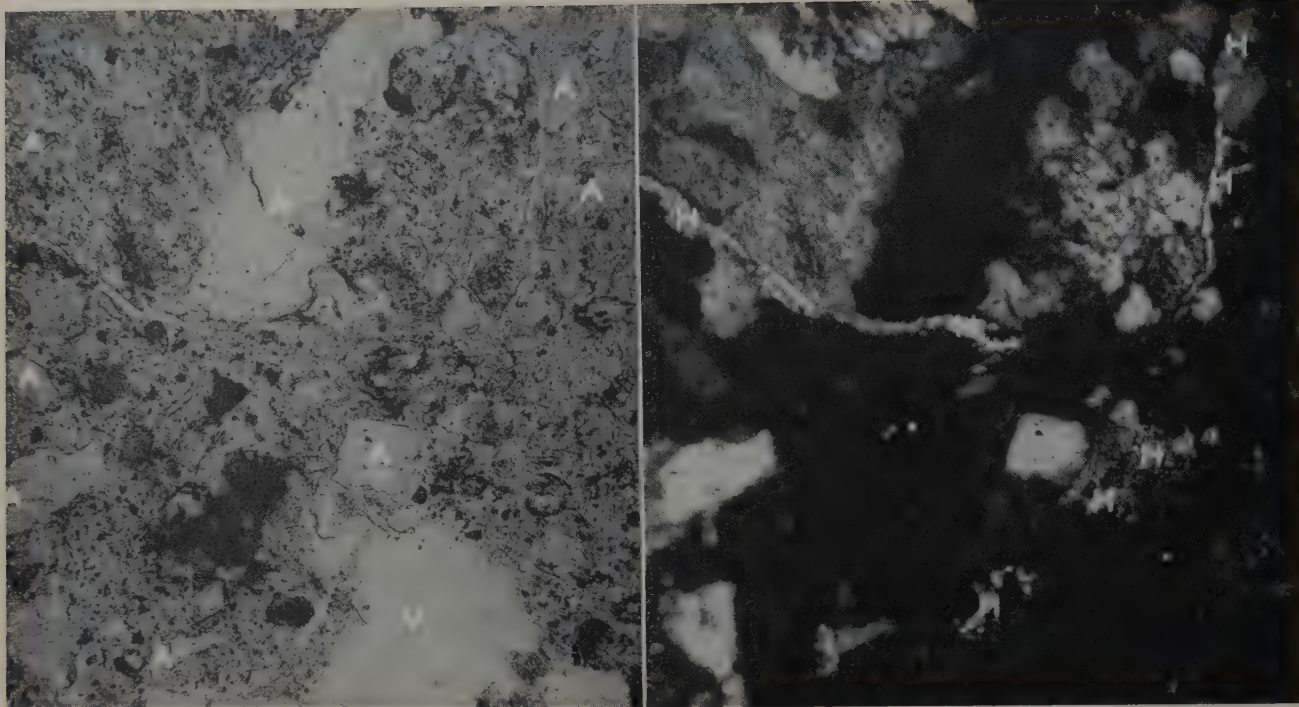


Fig. 4—Dam D, interior mass concrete containing diorite aggregate, X165, plane light (left), and crossed nicols (right). At left are shown voids (V), aggregate (A), and dark residual cement grains. In the right photomicrograph, the calcium hydroxide crystals are large, there is very little aggregate smaller than 25 microns, and there are larger areas of uninterrupted gel. Compare with Figs. 2 and 3 for differences with limestone aggregate concrete.

in dense rock; it gave a durability factor between 75 and 80.

While there were not enough aggregate combinations tested to make correlation analysis useful, the degree of association between type of clay mineral and durability factors above and below 80 could be computed by use of the chi-square test.¹² This calculation indicated that the probability of chance occurrence of the association between swelling clay and durability factor below 80 was less than 2 in 100.

Several examples of failure or distress in service of argillaceous limestone aggregates in which the major factors appear to have been the presence of swelling clay in limestones of high absorption were described by G. F. Loughlin.¹³ He investigated a number of limestones from Pennsylvania, Illinois, Iowa, and Kansas which had either poor service records or peculiar test results. All these rocks were argillaceous limestones, and all were shown to contain more than 5 pct of montmorillonoid clays. Variations in amount of clay, size and distribution of clay pellets, absorption, and distribution of permeable bedding planes and fractures appeared to explain the differences in performance between rock from different beds in the same quarry and from quarry to quarry. Small differences in amount and arrangement of clay accounted for differences in test results and in service.

The data so far available indicate that 5 pct or less of kaolin clay in dense limestone may not injure the performance of the limestone as aggregate, but that similar percentages of montmorillonoid clays in limestone with 1.5 to 2 pct absorption may produce a noticeable effect on durability as determined by accelerated freezing-and-thawing of concrete, and 10 to 20 pct of montmorillonoid clay in limestones which have 2 pct or more absorption, used in pavement laid on poorly drained foundations, may

cause the pavement to fail. No information has been located about the effects of some of the other clay minerals (illites, chlorites, attapulgite) that are recorded or possible constituents of carbonate aggregates. Information is also lacking on the mineral composition of shales interbedded with limestones, as that composition affects the performance of the aggregate. Not all these shales are eliminated in aggregate processing but persist especially in the smaller sizes of the finished coarse aggregate. Some cause pop-outs in freezing and thawing test beams and in concrete structures, but some do not.

Dolomitic limestones and dolomites have not been as well represented as limestones among the carbonate aggregates tested at the Waterways Experiment Station. One group of six dense calcitic dolomites was examined and tested during an aggregate survey made for a dam in Virginia. Four of the six were quite pure calcitic dolomites. One contained some cherty and some sandy dolomite as well. One was argillaceous dolomite, but the clay was not identified. All had less than 1 pct absorption. When each was tested in freezing-and-thawing of concrete in a mixture containing manufactured sand from the same source, all had durability factors of 85 or more, except one pure sugary-textured highly fractured dolomite.

Secondary silica in the form of chert or quartz or opal may affect both production and use of a carbonate aggregate. Vein quartz, like detrital quartz, increases the resistance to crushing of a carbonate rock and raises its thermal coefficient. Chert has a similar influence on crushing characteristics, particle shape, and thermal properties, and in addition is sometimes potentially reactive with the minor alkalis in portland cement. Opal is apparently much less common than chert, but opal is the major constituent of the Californian rock usually called *siliceous magnesian limestone* in the literature of alkali

reaction,¹⁴ a dolomitic opaline shale like that forming large concretions in the Monterey formation.¹⁵ This peculiar rock is unique. It is distributed in a restricted area and is not what is ordinarily called magnesian or dolomitic limestone. Three railroad structures in southern Ohio and Indiana containing limestone and dolomitic limestone coarse aggregate and natural fine aggregate are believed to have deteriorated because of reaction between chalcedonic chert and opal, as constituents of the fine aggregates, and the minor alkalis in the cement.¹⁶ The limestone coarse aggregates are not involved. Two of the structures were built in 1901. The age of the third was not given, but the youngest structures examined in this group of 13 were built in 1927. Each had some features suggesting poor construction practice, poor design, or poor drainage.

Four additional examples of alkali-aggregate reaction in field concrete containing limestone or dolomite as coarse aggregate have been discovered or confirmed as the result of the examination of samples by the writer. They have common factors suggesting that the alkali-aggregate reaction in these cases may be a secondary consequence of other deterioration in the concrete, rather than a major or a primary cause of deterioration. In three cases chalcedonic chert and glassy volcanic rock fragments, constituents of the fine aggregates, were the reactive ingredients, and there were no signs of reaction by the coarse aggregate. In the fourth, chalcedonic chert nodules in the limestone coarse aggregate had reacted, but this was the most porous and permeable concrete ever examined at the Waterways Experiment Station, and it was exposed to severe freezing-and-thawing and alternate wetting and drying. All the structures were in regions with fairly severe winters, and all were non-air-entrained concrete. No distress was recognized until the structures were more than 15 years old, and there was evidence of poor construction practice or high water-cement ratio or both in three. All contained smooth-surfaced rounded natural sands high in quartz.

The examples so far discussed to illustrate effects of variations in properties of carbonate aggregates with changes in composition have been concerned with variations among several groups of carbonate aggregates: limestones and dolomites, argillaceous limestones and dolomites, and cherty limestones. The last group of examples will illustrate variations in the properties of concrete resulting from differences between some carbonate aggregates and some non-carbonate aggregates.

In 1952 the writer had the opportunity of examining 10-in. diam diamond drilled cores of interior and exterior mass concrete from five Corps of Engineers dams, as part of a study of causes and methods of controlling cracking in mass concrete structures.

The fine and coarse aggregate used in the concrete in three of the dams was manufactured from limestone. In one it was manufactured from granodiorite, and in the fifth it was manufactured from quartz diorite. Each dam was represented by exterior and interior cores, and information was provided on the location in the structure from which the cores were taken, the age of the concrete when the cores were drilled, and the mixture proportions and aggregate gradations. Four of the aggregates had previously been tested at the Waterways Experiment Station, so that results of the standard physical tests, of freezing-and-thawing tests of concrete specimens, and of petrographic examinations

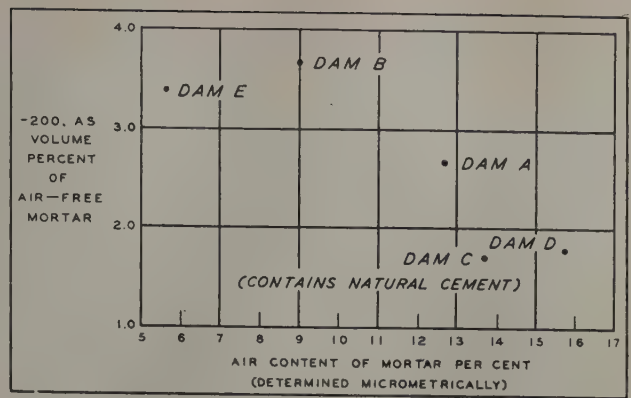


Fig. 5—Relation of average air content of mortar and amount of material finer than No. 200 sieve in aggregate.

were available. All the cores were examined macroscopically, the coarse aggregate content and air content of the hardened concretes were determined, sawed and broken surfaces were examined at low magnifications, and many thin sections of the concrete were prepared and examined.

The three limestone aggregates were not extreme variants in the limestone group, but they did differ considerably in structure, texture, and composition. One was relatively pure calcitic limestone, on the average finer-grained and more even-grained than the other two. The second was oolitic and fossiliferous limestone grading into sandy and shaly limestone. It ranged in grain size from very fine-grained to coarsely crystalline. The third contained about 25 pct chert, was the most cohesive of the three, and was moderately fine-grained. The two igneous aggregates were medium-grained slightly foliated granodiorite and moderately coarse-grained quartz diorite. The granodiorite was not altered but had some internal microfractures. The diorite was altered but was thoroughly interlocked by intergrowths of feldspar and quartz in the groundmass.

Comparisons of thin sections of the concretes made with limestone aggregates and those made with siliceous aggregates revealed several conspicuous differences between them. One was the difference in crystal size of the calcium hydroxide, which is one of the products of cement hydration, between the concretes with limestone aggregate, Figs. 2 and 3, and the concretes with siliceous aggregate, Fig. 4. Those with limestone aggregate had relatively small calcium hydroxide crystals, and those with siliceous aggregates, relatively large ones. This general relation, observed in all these cores, has also characterized all the comparisons made so far between undeteriorated concretes with limestone aggregates and comparable concretes with siliceous aggregates. It was also found between concretes of the same age made with the same cement and what was intended to be the same gradation of fine aggregate. The second consistent difference between the concretes with limestone aggregate and the concretes with siliceous aggregates was the difference in amount and particle size distribution of material finer than the No. 200 sieve in the aggregates. (Compare Figs. 2 and 3 with Fig. 4.) The difference in crystal size of the calcium hydroxide and the difference in abundance and size of the aggregate passing No. 200 implies another point illustrated by the photomicrographs: there are larger areas of apparently amorphous hydrated cement gel uninterrupted by crystalline particles in the concrete with siliceous aggregates.

This observation correlated with the tendency to crack in the cores from which the sections were made. When sawed surfaces of the concretes were examined at low magnification after several months exposure to air, all the limestone-aggregate concretes had more cracks per 100 sq in. than either of the siliceous-aggregate concretes.

The consistent difference in amount and particle-size distribution of the aggregate finer than the No. 200 sieve depends on differences in physical properties between the mineral constituents in the two groups of aggregates. The percentages passing No. 200 in the fine aggregate gradations reported showed an average of 3.5 pct in the two containing siliceous aggregates, and 4.8 to 8.1 pct in the three containing limestone. The actual quantity of -200 in the concretes included the reported amount, as well as an additional amount consisting both of that adhering to coarse aggregate and not allowed for and that produced by wear between the point where samples were taken for sieve analysis and the final location of the concrete.

In the crushing of brittle crystalline solids, the amount of new surface formed per unit of energy input increases as the hardness on Moh's scale decreases.¹⁷ Quartz has a hardness of 7; the feldspars range from 6 to 6.5; calcite has a hardness of 3. The same expenditure of energy therefore yields about 1.8 times as much new surface on calcite as on quartz and feldspar.¹⁷ If all the aggregates were similar in original grain size and interlock and were subjected to the same amount of abrasion during batching and mixing, the amount of new surface produced on the limestones should be about 1.8 times that produced on the granodiorite and quartz diorite, and the amounts of -200 mesh should be proportionately different. As a matter of fact, the limestones were of finer grain; it was difficult to compare the degree of interlock between the limestones and the diorite and granodiorite.

There were two other features of these concretes related to the amount of -200 in the concretes. Like all portland cement concrete placed by conventional methods, all these cores showed some evidence that the mortar had bled water and subsided under the larger coarse aggregate before set. However, the limestone-aggregate concrete bled less and subsided less under the aggregate than the siliceous-aggregate concretes. These observations may be explained as due to the effect of additions of material of high specific surface in reducing bleeding, and the greater effectiveness in this respect of pulverized limestone as compared to pulverized quartz of similar specific surface.¹⁸

The additional effect of the larger amount and smaller average particle size of the -200 limestone was a reduction in amount of entrained air in mortar, Fig. 5, and a reduction in maximum and median size of the individual air voids. A given air content in voids of small size should give more frost resistance than the same air content in voids of larger size.¹⁹

Summary

It is believed that the examples that have been presented illustrate the following points:

1—The carbonate aggregates have a wide range in composition, from calcitic limestones and pure dolomites to argillaceous and arenaceous limestones and dolomites. Each of these types may also vary widely in texture and structure.

2—The variations in composition, texture, and

structure result in wide variations in physical and chemical properties, and potential or actual suitability for use as concrete aggregate.

3—Carbonate aggregates that contain even small amounts of swelling clay and that also have fairly high absorption may give poor service, particularly in concrete exposed to severe weathering or frequent wetting and drying. However, dense argillaceous limestones in which the clay is not swelling clay may give good service. Determination of the clay mineral or minerals and the amounts in which they are present offers the most satisfactory basis for a decision.

4—Calcite and dolomite, the major carbonate minerals, are considerably softer than the feldspars, quartz, and the ferromagnesian minerals that are the principal constituents of most of the other sorts of rocks used to produce crushed aggregates for concrete. As a consequence, more material passing the No. 200 sieve is likely to be produced in processing and handling carbonate rocks than would be produced from rocks of other types.

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Recharging Ground Water Reservoirs With Wells and Basins

by M. L. Brashears

IN the last 15 years industrial use of ground water has more than doubled, and in 1951 amounted to 5 billion gallons per day. A similar sharp increase in the utilization of ground water for irrigation and public-water supply occurred in the same period. In many areas rapid increase in withdrawal from wells has taken place almost entirely unhampered by regulatory control and with little or no integration of effort. As might be expected, the chief interest in many regions has been maximum production rather than sustained perennial yield. As a result, widespread depletion of underground reservoirs and deterioration of the quality of the water stored in them has taken place in many areas, even though total pumpage in the United States is far below ultimate potential. Of even more concern is the fact that excessive withdrawal has drawn salt water into the reservoirs beneath many heavily populated centers along the Atlantic, Gulf, and Pacific coasts, causing costly abandonment of pumping plants.¹

Many hydrologists expect that consumption of water will rise rapidly in the near future, and some predict that industrial requirements will more than double in the next decade.^{2, 3} Thus it appears likely that the draft on many already heavily pumped underground reservoirs will be greatly increased and the search for additional sources of usable ground water intensified in years to come. In view of this, industry as a whole will be forced more and more to recognize the potentialities and limitations of ground-water reservoirs and to utilize them more effectively to prevent costly water shortages and disruption of production. Through painful experience, some industries are already well aware of the need for effective water utilization, and have managed individually or through joint effort to check trends threatening to deplete underground reservoirs completely or to impair the quality of the water. Various remedial measures have been used to bring about successful management of local or regional ground-water resources. Of these, replenishment of aquifers by recharge wells or basins has played an important role in overcoming some ground water problems.

Artificial recharge of underground reservoirs by water spreading has been practiced successfully in the United States for many years. In the West it has become an important method of salvaging flood run-off for irrigation of crops and maintenance of public water-supply reserves, and it is used to some extent in parts of the East. Artificial recharge by means of wells, on the other hand, is a relatively new development. Until recently it was employed in only a few areas, principally along the East coast. For the last few years, in the ever increasing search for additional water supplies, industry has had greater recourse to this method. Utilization of

recharge wells to control the temperature and quality of underground water supplies is also being considered seriously.

Operation of recharge wells, like water spreading, is governed largely by local conditions. It requires water relatively low in turbidity, whereas in some areas water spreading has been used successfully with water of high turbidity and silt content. However, water spreading must be employed in large areas and can be carried on effectively only where aquifers crop out at the surface. Recharge wells can be used in limited space.

Recharge wells are similar to production wells except that the water flows in the opposite direction. Thus any water-bearing bed that will yield water to wells may be recharged by wells. Often, however, the water available for recharge is of a different character and temperature from that existing in the ground-water reservoir and if transmitted directly underground from a recharge well to a production well might require expensive or difficult treatment before it could be used.

Fortunately the physical characteristics of reservoir beds, which control the movement and behavior of ground water, are generally not homogeneous. Moreover, the movement of ground water is very slow because of the frictional resistance of the reservoir beds. By taking full advantage of hydrologic and geologic conditions, it is therefore possible in many instances to bring about favorable changes of temperature and dilution as the water moves from the recharge wells underground to the production wells. Furthermore, if the natural quality or temperature of ground water is unfavorable for industrial purposes, recharge wells may be used to introduce water of more favorable quality or temperature into the ground-water reservoir.

When water is discharged into a recharge well, the head in the well is increased. Because of this, a cone of elevation is produced on the water table or the artesian pressure surface in the area surrounding the well. The cone of elevation is similar to the cone of depression produced around a pumping well except that the apex of the cone is above the water table or artesian pressure surface. Thus if a recharge well and a production well tapping the same water-bearing bed are close together, as would be the case at many industrial plants, some of the water discharged from the recharge well would be drawn into the production well within a short time. Under such conditions it is apparent that water of unfavorable temperature and chemical characteristics should not be used for recharging.

The more important ground-water reservoirs in the United States often consist of alternating layers of impermeable beds and porous material that will yield water readily to wells. Physical characteristics of individual beds in a ground-water reservoir may not persist over great distances, the impermeable layers grading into beds that will yield large quantities of water. Thus the water-yielding material in underground reservoirs, whether large or small,

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Discussion on this paper, TP 3609H, may be sent (2 copies) to AIME before Dec. 31, 1953. Manuscript, January 6, 1953. Chicago Meeting, October 1952.

often consists of a maze of interfingering and overlapping channels of permeable material of varying thickness through which ground water moves on its journey from intake area to discharge area, or from recharge well to production well.

Because of this irregularity in most ground water reservoirs it is often possible to pump water from one of several water-bearing beds and to recharge another bed separated from it hydraulically, so that movement of water between the two wells is prevented. Movement of ground water is influenced by every minute irregularity of the reservoir beds. Even thin impervious beds or variations in permeability within an individual water-bearing bed may greatly influence the direction of movement of ground water. Deposits of sand and gravel are more or less horizontally stratified. This often results in very thin layers of fine sand, silt, or clay, the presence of which would not be recognized during the drilling of a well. The vertical permeability of deposits of this kind is much less than the horizontal permeability, the thin layers of relatively impervious material greatly retarding the vertical movement of water. With such deposits there often is a great advantage in having the recharge well of different depth from that of the production well.

All water-bearing beds serve not only as reservoirs but also as conduits to carry water to wells from the areas where it enters the beds. In this respect ground-water reservoirs differ from lakes and surface reservoirs in that they offer great resistance to movement of water through them. Serious water shortages may develop in ground-water reservoirs not only because of insufficient storage capacity or natural replenishment but also because the slow movement of ground water through the reservoir beds, even under high hydraulic gradients, cannot meet the demand of pumping wells. This important characteristic of water-bearing beds is important in all problems of artificial recharge.

No two ground-water problems are exactly alike, and each may result from a complex set of natural and man-made conditions. In some instances it may be possible to remedy the situation by means of artificial recharge, but in others artificial recharge may cause additional problems. Careful evaluation of conditions causing the problem should be made if waste of time and money is to be avoided. Spacing of wells is of major importance. Where wells are closely spaced, as in highly developed areas, artificial recharge practiced at only one industrial plant may produce only minor beneficial results. Joint effort, however, may bring about a satisfactory remedy for all. On the other hand, where wells are spaced great distances apart the entire benefits of artificial recharge may be realized by the operator.

Broadly speaking, industry may obtain water for artificial recharge from the ground-water reservoir being recharged or from some outside source. The latter may be derived from a municipal supply system, nearby lakes and streams, or even from wells tapping another ground-water reservoir. It has been demonstrated on Long Island and elsewhere that if properly treated, industrial waste water, storm water, and sanitary sewage may be used for recharging. Selection of source would be governed by many factors, chief of which would be temperature, turbidity, and chemical character of the water and the distance to the source.

There are numerous situations where artificial recharge wells could be profitably used by industry

for cooling large quantities of water during the warmer months or for controlling temperature and chemical character of water in the underground reservoir. Under some conditions, recharge wells may be used profitably for disposal of waste water and the draining of water-logged land.

Usually the amount of water stored in an underground reservoir is equal to annual replenishment during several years or more. Because of this, water may often be pumped for short periods of time at rates considerably greater than the rate of perennial replenishment without creating serious problems. However, each period of heavy withdrawal must be followed by a period of little or no pumping, or the water must be replaced by artificial recharge. Such water-management procedure has been carried out successfully at several places in the United States. Perhaps one of the most striking examples of the use of artificial recharge to prevent exhaustion of available supply took place during World War II in Louisville, Ky, where an acute water shortage developed because of excessive withdrawal. Several industrial well fields were pumped at a rate of 60 million gallons daily, about twice the amount of natural replenishment. Late in 1943 the yield of some wells had dropped far below requirements and prompt action was needed to prevent further decrease in yields and to maintain an adequate supply of water with a summer temperature no higher than 60°F. Through joint effort, the industries arranged to use Louisville municipal water for several months during the winter and spring of 1944. During the same period the depleted underground reservoir was recharged by the pouring of cold water from the city supply into industrial wells at the rate of about 1000 gal per min. According to Guyton⁴ this increased the potential yield of the area during the entire summer of 1944 by about 1000 gal per min and curtailment of production was prevented.

Similar beneficial results have been obtained at an industrial plant in the Binghamton, N. Y., area, where gradual increase in pumping from a small underground reservoir at the plant site caused a large decline of water levels. Additional wells were drilled several thousand feet from the plant near the Susquehanna River, the major source of the ground water. It was soon observed that although there was a lag of several months the temperature of the water pumped from the newer wells was governed by changes in temperature of the Susquehanna River.

During the late summer and early fall, temperature of the water from the newer wells rose above the operating maximum of 55°F. A schedule of recharging has remedied this situation. Cold water taken from the wells near the river during the off-peak season is pumped into the cone of depression around the plant, where it is held in storage until needed the following season. Under similar conditions elsewhere it is probable that cold water pumped from off-site wells or streams could be transported economically over long distances.

Canton, Ohio, affords another example of the successful operation of recharge wells in utilizing an underground reservoir for temporary storage of excess run-off. According to Kazmann⁵ one of the Canton well fields lies over a buried valley containing two gravel aquifers separated by a layer of clay. The upper aquifer, although readily recharged by streams which cross it, is relatively thin and has

insufficient storage capacity to furnish the desired supply during periods of low stream flow. The lower aquifer, although comparatively thick, has no clearly determinable source of recharge and would be quickly exhausted if pumped heavily. Three horizontal collector wells were sunk along streams crossing the area. These collectors tap both aquifers and allow water to flow continuously from the upper to the lower, where it is held in storage until needed. The combined rate of recharge of the three collectors is 8 million gallons per day. One of the collectors is equipped with pumps and produces 10 million gallons per day from the lower aquifer.

As a result of control by the state government, more recharge wells have been drilled on Long Island, N. Y., than in any comparable area in the country. In 1933 it was discovered that excessive pumping in the previous decade or two had lowered the water table below sea level in an area of more than 40 square miles in Brooklyn and that sea water was moving into this area. Rough estimates indicate that approximately 100 billion gallons of ground water were mined during the period in which pumpage exceeded natural replenishment.

To protect public water supplies on Long Island from further over-development the state legislature passed a conservation law requiring the approval of the State Water Power and Control Commission before a well was drilled having a capacity greater than 100,000 gal per day. Since passage of the law the Commission has required that water pumped from new cooling and air conditioning wells be returned to the aquifer from which it is withdrawn. More than 300 recharge wells have been installed since 1933.⁶ About 10 pct of these are industrial wells operated the year round. The remainder are operated only in the warmer months of the year.

The experience gained on Long Island during the last 20 years clearly shows that recharge wells can be operated successfully in unconsolidated deposits (the most productive aquifers in the heavily industrialized areas of the nation) under almost all geologic and hydrologic conditions. However, construction methods must be proper for the physical characteristics of the aquifers and the water used for recharging should be relatively clear. Moreover, hydrologic conditions at some locations interpose limitations. For instance, one limiting factor is depth of the water table below surface. If the water table is near surface the build-up of head in the recharge well must be relatively small. Under such conditions it is generally possible to create a draw-down in a pumping well greater than the maximum build-up that can be attained in a recharge well. Thus if other conditions are approximately equal, maximum rate of recharge may be considerably less than the rate of yield of a supply well. Under such conditions it is necessary to construct more than one recharge well to receive the discharge of a single supply well. Despite these limitations carefully constructed recharge wells, some with capacities of more than 1000 gal per min, have been successfully operated on Long Island for periods as long as 10 years without repair or rehabilitation. Even so, many operators consider it good practice to clean out or redevelop recharge wells every year or two.

Different kinds of recharge wells have been constructed on Long Island, but most are of two basic types, the so-called dry type and the so-called wet type. The former consist of perforated cased holes ending above the water table; the latter are

screened below the water table or in an artesian formation. Available records indicate that about 75 pct of the recharge wells so far constructed on Long Island are screened below the water table. Selection of the most suitable type for a particular problem depends on geologic and hydrologic conditions.

Each type of recharge well has certain inherent advantages. It is commonly believed that the wet type is more successful. In the dry type of well, warm water may be discharged at a considerable distance above the water table, permitting some dissipation of heat as the water passes downward to the zone of saturation. This kind of well is usually finished with a section of large diameter pipe into which slots have been cut. The wet type of well is generally equipped with a standard well screen. The dry type of well, which is also shallower, is therefore less expensive to construct. However, excessive clogging is reported to take place in some of these wells. Some investigators believe the clogging is caused by release of dissolved gases as water leaves the perforated pipe and by contact of water with oxygen in pore spaces of the unsaturated sediments. There are no adequate means of reconditioning these wells, whereas the same methods used to re-develop supply wells, i.e., surging, bailing, and chemical treatment, can be applied to the wet type of recharge well. On many occasions supply wells have been used as recharge wells, continuously, or alternately as supply well and recharge well.

Depending on the type of cooling plant, the temperature of water returned to the ground on Long Island ranges from 2° to 20° higher than the temperature of water pumped from the ground. It has been shown that the return of warm water to the ground causes a regional rise in ground water temperature.⁷ At some installations this rise has lowered the advantage of using ground water for cooling purposes by causing a decrease in plant efficiency. However, appreciable areal rise of ground water has occurred only in the vicinity of centers of considerable warm water recharge. Where wells are not closely spaced or are vertically separated by impervious layers there has been no serious rise in ground water temperature. Moreover, it has been observed that ground-water temperature, where raised by recharge, will generally decrease rapidly if the return of warm water is discontinued. Where an aquifer contains comparatively thin impervious layers it has been observed that there is little or no rise in the temperature of the water pumped from a supply well.⁸ At a large ice plant near tide water in Brooklyn, a supply well and a recharge well, about 100 ft apart, were screened in the upper part of a gravelly aquifer containing two thin layers of clay. Shortly after recharging operations began, the temperature of the water pumped from the supply well began to rise and soon reached a temperature 20° higher than at the start of operations. The supply well was then deepened and screened in the lowermost gravelly layer. Operations were resumed and have been continued for 10 years without rise in temperature of water pumped.

But pumping from the lower gravelly layer at this plant upset another natural hydrologic balance. The supply well, when screened in the uppermost layer, produced water of low salinity. Shortly after the well was deepened, pumping began to draw sea water into the area. Eventually the chloride content in the pumping well leveled off to about 5500 parts per million, or about one fourth that of sea water.

Apparently water of this character was acceptable for compressor cooling at the ice plant, although it would not be satisfactory for many other purposes, but if the shallow well had been used as the supply well and the deeper well as the recharge well, fresh water of natural temperature probably could have been pumped continuously.

Operational experience at this ice plant illustrates how recharge wells may be used to control the temperature and chemical character of ground water if advantage is taken of favorable geologic conditions. This experience also suggests other ways in which recharge wells may be used to control the temperature and quality of ground water. One of the more obvious uses would be to lessen the danger of sea water invasion by introducing water into an aquifer at places between pumping wells and tide water, thus developing a fresh water divide between the pumping wells and the sea. Successful experimental results of this kind have been obtained by Laverty⁹ along the Pacific coast west of Los Angeles.

Another allied use of recharge wells successfully practiced in Brooklyn and at El Paso, Texas, is the discharge of fresh water into a salted aquifer, from which it is later pumped at times of peak demand. Thus underground storage is utilized in place of surface storage. Artesian aquifers have been used for day to day storage on Long Island and for season to season storage at El Paso. In Brooklyn during off-peak periods, well water from public supply mains has been discharged into production wells, screened in salted aquifers. A few hours later during periods of peak demand the fresh water was pumped from storage. It is reported that about 20 pct of the fresh water injected into the aquifer was lost because of mixing with salty ground water.

At El Paso, Texas,¹⁰ prior to 1948, heavy demands for public supply resulted in pumping rates in excess of the recharge to the aquifer, and the chloride content of the ground water was steadily increasing. To offset this condition, treated Rio Grande River water has been pumped into the aquifer for a 90-day period during four of the last five winters.

Such water management schemes cannot be effected successfully by hit or miss drilling of wells, but, as shown by elaborate experimentation at El Paso, would have to be founded on sound design and construction and careful operation.

Hydrologic trends on Long Island in the last two decades clearly indicate the danger of pumping water from wells without observing its effect on the natural balance previously existing. Prolonged delay in effecting legislation to curb excessive mining of ground water in Brooklyn, due largely to complete ignorance of the rapid lowering of water levels, has caused abandonment of all public supply wells in Brooklyn, a multi-million dollar loss. Although state control has checked additional ground water development in Brooklyn since 1933, pumpage from old wells has continued. As a result, water levels continued to decline from 1933 to 1942. Thereafter, water levels recovered slowly, but since the water table was still below sea level in much of Brooklyn, sea water continued to move into the area. Finally in 1947 the only remaining public-supply system serving ground water, a plant of 30 million gallons daily, was abandoned because of the invasion of sea water. Since then, and unfortunately only after irreparable damage had occurred, water levels have recovered rapidly.

Sometimes the aftermath of tragedy is accompanied by unexpected beneficial developments. Mismanagement of ground-water resources of Brooklyn has highlighted the possibility of using the underground reservoir in Brooklyn as an additional dry weather source of water for the New York City municipal supply system. Wet weather run-off, which now spills over the city's upstate dams, might be discharged readily into the regional cone of depression now existing in Brooklyn. Once refilled, there would be sufficient water in storage which could be pumped from the same wells to satisfy the need of the entire city for approximately 100 days, although one important factor may make the realization of this scheme difficult. After the water table was drawn below sea level earlier in the century a number of subway tunnels and other substructures were constructed in Brooklyn. It may not be possible, therefore, entirely to refill the underground reservoir without serious flooding problems.

In view of expanding industrial requirements, the search for additional sources of usable ground water will undoubtedly be intensified in the next decade. Because of this, and because much of the usable ground-water supply in the vicinity of many heavily industrialized areas in the country has already been appropriated, there is good reason to believe that many industries will be faced with costly water shortages unless wasteful utilization is replaced by intelligent water management. Much of the water now pumped from wells by industry is not consumed but is used as a heat exchange medium or a cleansing agent. Large quantities of water now discharged into rivers or the ocean could be profitably re-used if it could be economically cooled or held in storage for peak demand or dry weather use. Successful experience during the last decade has demonstrated that modification of the temperature and quality of waste water can be economically attained by recycling plant effluent through ground-water reservoirs. This, of course, does not imply that all industrial ground-water problems may be solved by the use of recharge wells. However, it is strongly evident that this method of conserving water supplies has many fruitful applications and that artificial replenishment of ground water reservoirs by means of recharge wells should be given careful consideration whenever industrial ground-water problems arise.

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Process Development and Practice of the Potash Division of the Duval Sulphur and Potash Co.

by G. E. Atwood and D. J. Bourne

The new potash refinery of Duval Sulphur & Potash Co. produced its first tonnage of muriate of potash in November 1951. Full capacity was attained in early 1952. Excellent extraction efficiency is being obtained by the process, which is basically froth flotation. System temperature control is practiced and has proved to be of great metallurgical value through high potassium recovery and low reagent cost.

WHEN Duval Sulphur and Potash Co. recently entered the established potash industry, the vast amount of pioneer work essential in an entirely new endeavor was unnecessary. Thus it was possible to screen and assemble proved techniques into a highly efficient process based on the combined knowledge of the industry. It is hoped that the refinements Duval has added to these cumulative

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Discussion on this paper, TP 3672H, may be sent, (2 copies) to AIME before Jan. 31, 1954. Manuscript, Jan. 15, 1953. Los Angeles Meeting, February 1953.

achievements will contribute to the rapid advance of the potash industry and related enterprises.

The process selected utilizes as raw material sylvinite ore typical of the Permian Basin. This ore is principally comprised of interlocked crystals of potassium chloride, about 40 pct, and sodium chloride. Also there is as much as 1½ pct of dispersed clay slime which, because of its nature, highly complicates beneficiation of these ores. The refining method is basically flotation in which the mineral sylvite is floated away from the halite-clay gangue to produce a 60 pct K_2O muriate of potash product for the plant-food trade. It departs from established practice in the use of an improved reagent scheme, developed by the company's technical staff, and in

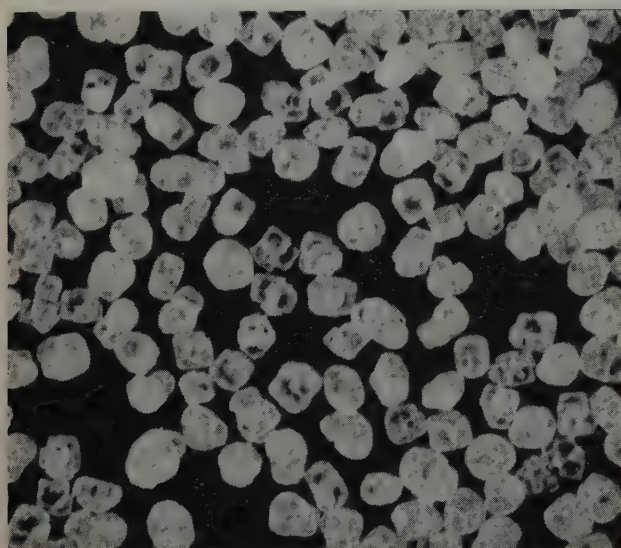
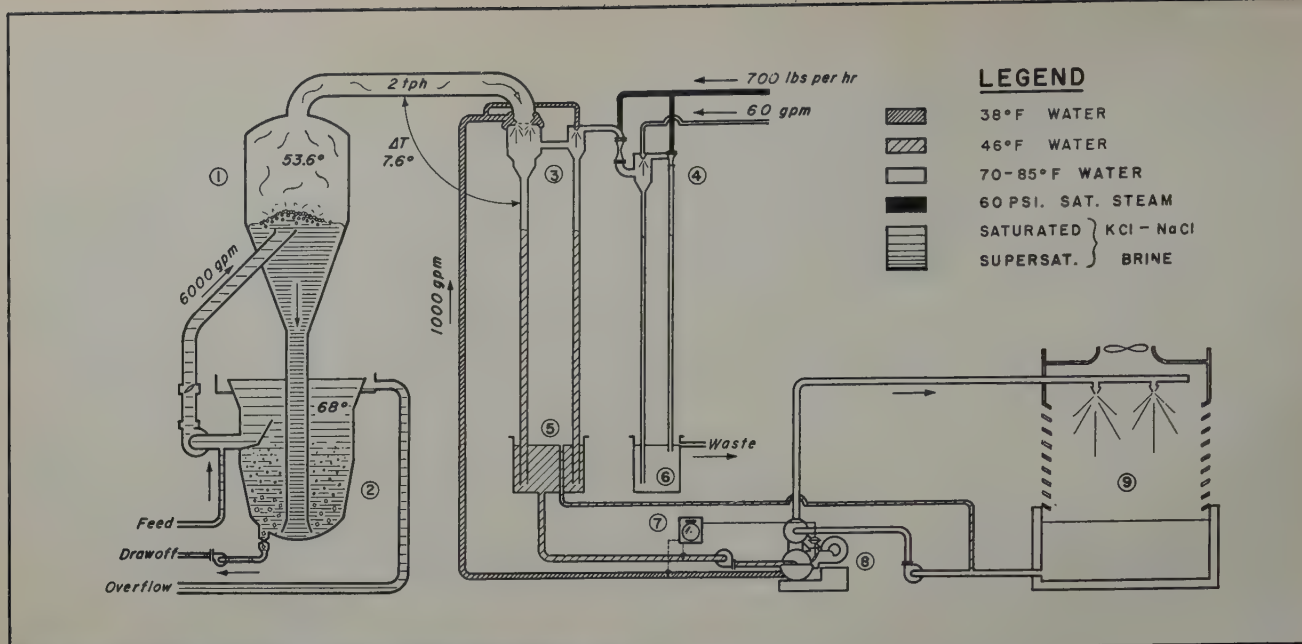


Fig. 2—The uniformity of crystals made possible by the Struthers-Wells Krystal vacuum crystallizer is illustrated above.

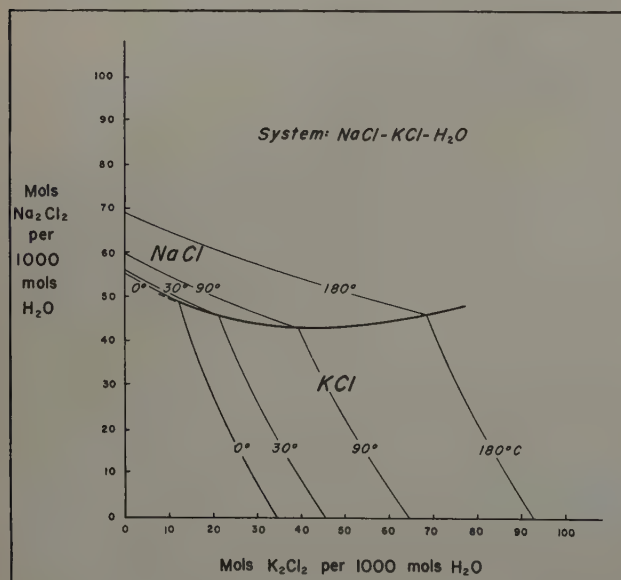


Fig. 3—Sylvite ore solubility data.

Fig. 1—Krystal flow diagram. Brine feed of 252 tph at 78°F enters the circulating line where it is pumped to the vaporizer (1). Following controlled vapor release, the supersaturated liquor passes to the suspension chamber (2). Crystal production of 2.3 tph is removed in a pulp with 7.7 tph of brine at the drawoff. Chilled brine overflow is 240 tph at 68°F. Vapor and non-condensable gas are respectively condensed and purged in a barometric condenser (3) and the two-stage ejector (4). Vessels (5) and (6) serve as the cold and hot wells. Chilled water for vapor condensation is provided by the Carrier centrifugal refrigeration machine (8), with refrigeration output controlled by the Foxboro temperature relation controller (7) actuating the suction damper to the compressor. Marley cooling tower (9) provides 70° water for condensing the refrigerant, Freon II.

the unique manner in which temperature control is practiced.

The obvious importance of temperature in beneficiation of soluble ores dates from early solution and crystallization practice. To a somewhat lesser extent, yet still highly significant, is its effect during collection by flotation reagents. Since the process chosen embodies both flotation and crystallization, system temperature control was incorporated as a basic feature. The physical scheme of control is that of removing a fixed amount of heat from the system at one point while introducing a variable quantity at another. Heat supply is varied to provide optimum temperature of the overall system and constitutes a balance with the effects of a widely varying ambient temperature on 1¼ million gallons of brine in the closed circuit.

Constant heat withdrawal is accomplished by a Struthers-Wells Krystal vacuum crystallizer. Vapor and non-condensable gases from the Krystal vaporizer are respectively condensed and purged by a chilled-water, multi-spray, barometric condenser and two-stage Schutte-Koerting ejector. Chilled water for vapor condensation is provided by a Carrier centrifugal refrigeration machine rated at 333 tons per day. A general flow diagram of the installation is seen in Fig. 1.

Selection of the Krystal to release heat from a saturated NaCl-KCl brine was based entirely on its inherent capacity to yield a product of desired physical quality. This is thought to be the first application of the unit on potassium chloride in

this country, and its efficiency is evidenced by an hourly production of 2.3 tons of +28 mesh high-purity product.

The Krystal superiority arises from the adroit manner in which the widely-known principles of crystallization are simply applied. These principles¹ are reviewed as follows in the light of their application to this unit. The driving force necessary for both crystal formation and growth is defined as supersaturation. The degree of supersaturation may be divided into the metastable and labile fields. The former describes a temperature concentration relationship where crystallization occurs only on seed or nuclei already present. The labile field is supersaturation above metastability wherein spontaneous formation of new crystals occurs without the stimulation of seed or nuclei. Design features of the Krystal² are such that solution feed is maintained within the metastable field where crystal growth can be controlled. This is accomplished by 1—providing proper circulation rates through the vaporizer to hold supersaturation to the desired degree and 2—maintaining a dense bed of crystals in the suspension chamber adequate to remove supersaturation so produced. It can be noted in Fig. 1 that the liquor movement is downward from the vaporizer and upward through the crystal suspension. A classifying action is effected by the ascending flow, and only the finished crystals of desired size approach the draw-off. Hence varying the rate of draw-off, within the limits of capacity, results in mesh control. The uniformity of product is apparent in Fig. 2. The crystals are conducted to the drying section of the plant, where they are mixed with flotation concentrate adding both physical and chemical quality to the product.

To refer again to Fig. 1, the data presented reflect typical operation and heat balance for the section. The aim in the design of the section was to maintain a plant brine temperature of from 20° to 25°C. This target seemed practical from the standpoint of year-round ambient temperature variation, corrosion considerations, efficacy of flotation reagents, and phase relations in the NaCl, KCl, H₂O system. To produce the relatively cold brine necessary to hold the plant circuit at 20°C the Krystal operates at from 0.3 to 0.4 in. Hg absolute in the vaporizer. Vapor temperature plus a boiling point rise gives the brine temperature indicated, and vapor temperature less the terminal drop across the condenser necessary to drive the reaction at the capacity desired will give the indicated chilled water return temperature. Overload and icing protection on the chilled water circuit is provided by the Foxboro temperature-relation controller. Simple, safe, and efficient refrigeration is supplied by the compact Carrier centrifugal compressor. It is equipped with a suction damper air-operated by the controller.

Application of crystallization to NaCl-KCl brines has been practiced for many years. The compatibility of process with system is apparent from a glance at the phase data. Fig. 3 is the system KCl, NaCl, and water.⁸ The solubility of potassium chloride decreases rapidly with temperature reduction, while sodium chloride varies but slightly over a wide temperature range and is in retrograde below 95°C.

Thus the saturated brines below 95°C lend themselves nicely to vacuum crystallization. In this range, slight increases in solubility tend to offset

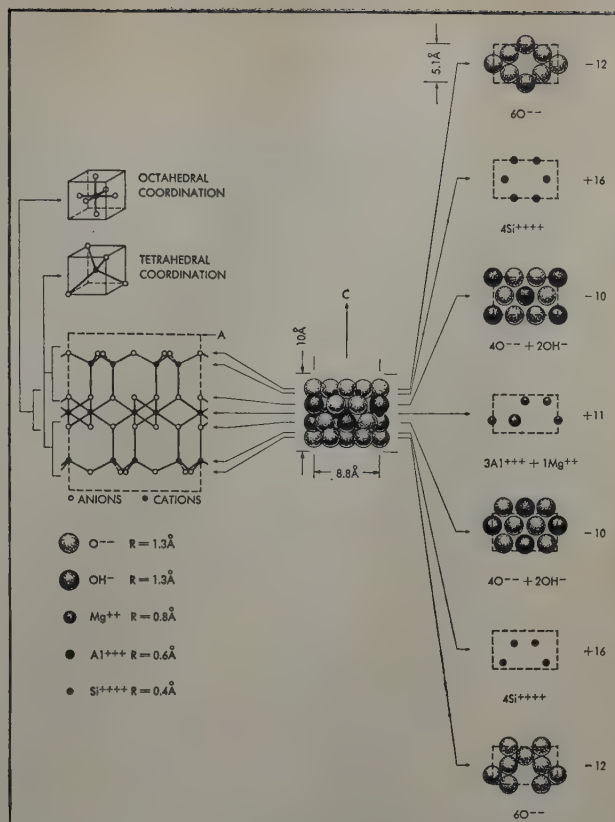


Fig. 4—The structure of montmorillonite 2 [(Al_{1.67}Mg_{0.33})Si₄O₁₀(OH)₂]. (After F. H. Norton, *Elements of Ceramics*, 1952.⁵)

sodium supersaturation caused by vapor removal. This salient feature assures sodium-free potassium chloride crystals. Throughout all ranges a high yield of potassium per degree of temperature reduction is apparent from the slope of the saturation curve.

On the basis of these factors the effect of heat applied to a suspension of ore constituents in brine can be predicted. Accordingly all control heat is introduced to plant tailings just prior to filtration. In this manner any solid phase potassium values that have escaped the flotation section are dissolved

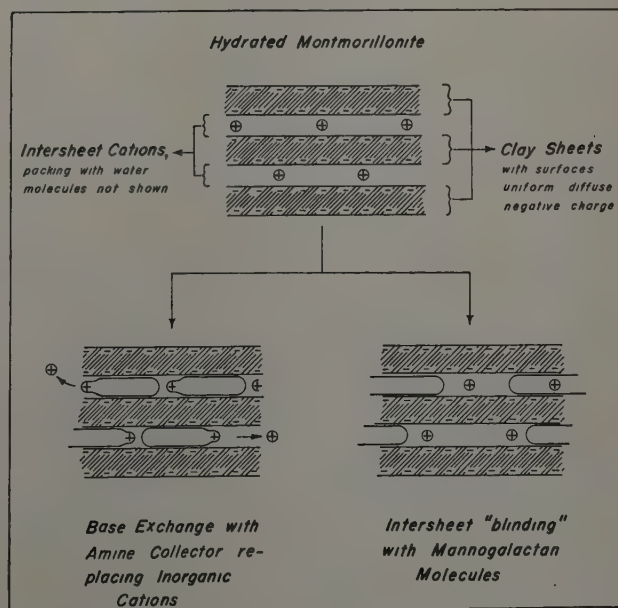


Fig. 5—The formation of montmorillonite complexes.

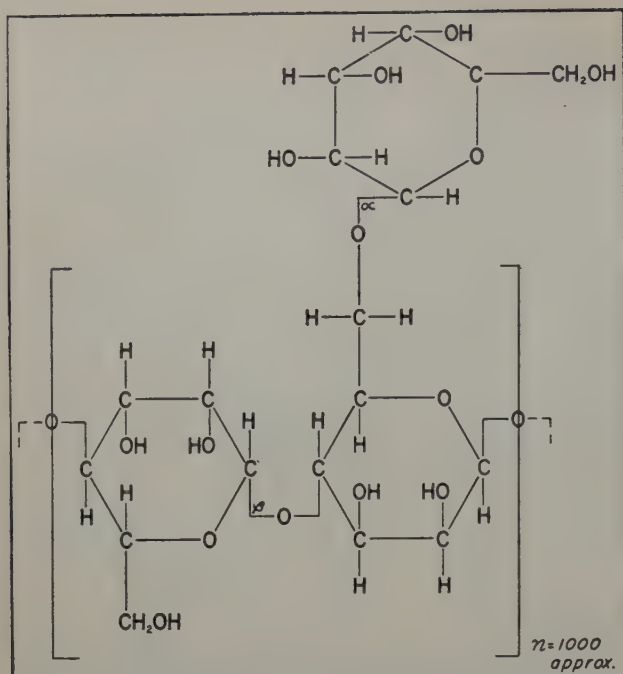


Fig. 6—The structural configuration of the unit cell of guar.

and are retained in the system for ultimate harvest in the Krystal.

Flotation recovery under these controlled temperature conditions is excellent and depends only slightly on secondary Krystal production for overall high process efficiency. A primary aliphatic amine salt manufactured from beef tallow is employed as the frothing and sylvite-collecting agent. The use of cationic agents in mineral flotation, though developed to a high degree, presents many complex problems with clay-bearing ores. To evaluate the adverse effect of clay slimes, it is necessary to have a clear understanding of their behavior and structural features.⁴

The clay associated with the Carlsbad ores has been identified as a member of the montmorillonite group. The approximate crystal structure of montmorillonite was suggested by Hoffman, Endell, and Wilm, and their work has been generously augmented by Hendricks, Bradley, McEwan, and others. The configuration generally accepted for the unit cell is shown in Fig. 4, outlined by dotted lines. The montmorillonite crystal is built from sheets of unit cells superposed one upon the other and held together by relatively weak forces. The oxygen-to-oxygen distance normal to, or separating the sheets, is variable from about 0.5 to 12Å and depends upon conditions of exposure.

The basic pattern in the unit cell is the four-high packing of oxygen and hydroxyl ions coordinated about three layers of cations. The upper and lower cation layers consist of silicon tetrahedrally coordinated with oxygen. The central layer is aluminum octahedrally coordinated with hydroxyl and oxygen.

Occasional substitution of lesser-valent cations in either of the coordination positions results in a residual negative charge for the unit cell. In montmorillonite, the residual charge arises in the main from substitution of magnesium, and sometimes iron, for aluminum in the octahedral coordination position. In Fig. 4 note Norton's substitution of one magnesium for an aluminum ion in the central cation layer.⁵ A net negative charge is arrived at by total-

ing the valence charges in the pictured cell. This electrostatic charge is balanced by the presence of replaceable intersheet ions and accounts for the high base exchange capacity peculiar to this particular clay. In addition, the montmorillonite adsorbs water and some neutral organic compounds, binding them with relatively weak forces, probably Van der Waals. Normally, montmorillonite has two or more intersheet layers of water molecules loosely held in a more or less regular pattern. These molecules are two-dimensionally fluid and offer little or no interference to the free movement of intersheet exchange ions. An accepted *ease of replacement* series for these intersheet exchange ions shows: large organic cations >, H^+ , Ca^{++} , Mg^{++} , K^+ , Na^+ ,⁶ thus the avid appetite of this clay for an aliphatic amine can be appreciated.

In hydrated montmorillonite, the intersheet spacing is sufficiently extended to pass readily large organic molecules which in turn displace the adsorbed water. Organic molecules on entering the clay crystal organize themselves for maximum utilization of the intersheet space, lying flat along the sheet surface and parallel to it, often in multiple layers. Hendricks⁶ reports that the total surface of water sorption by this clay is about 8 million square centimeters per gram, which is in the order of 50 times the observed external surface. Clay crystals are normally 0.2 to 0.5 square microns in the basal plane, and contain from 50 to 100 silicate layers. In relating these characteristics to typical plant operation, it is found in the flotation feed, in spite of efficient desliming, the ratio of active clay surface to KCl surface is greater than 2000 to 1.

A calculation based on ion exchange capacity of the montmorillonite of 0.85 me per gram indicates that this quantity of clay would consume 1300 lb of amine per day, or ½ lb per ton of ore feed. This figure can be supported from sorptive area and net charge per unit cell calculations. It is this coupling of the appetite and the capacity of the clay for the flotation collector that poses a major metallurgical problem in the process.

In as much as mechanical desliming methods are not perfect, the solution of the problem must arise



Fig. 7—Contrast of circuit clarity when using starch (left) and mannogalactan (right) as blinding agents.

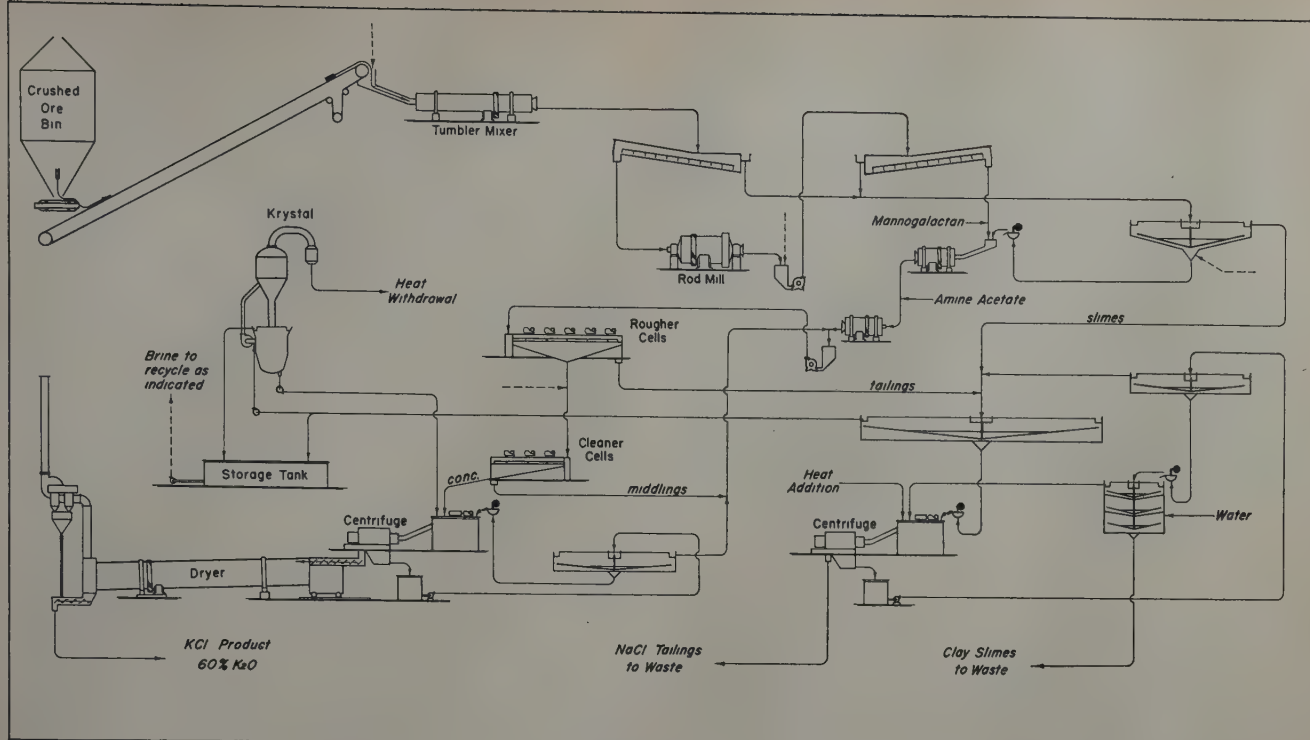


Fig. 8—The refinery flow diagram above shows the adaptation of the Krystal unit and the conditioning technique applied to mannogalactan.

from successful treatment of the unremoved clay. Effective treatment need not be permanent but must apply as long as the montmorillonite is in competition with the KCl crystal surface for the amine collector, i.e., while the flotation reaction is occurring. There appear to the authors at least three approaches to clay treatment:

1—Closing the sheets by irreversible dehydration with heat treatment. Heat-treated montmorillonite assumes a stable water-free configuration with less than 1 Å intersheet spacing. This contracted crystal structure could not then pass the amine radical in base exchange. This solution seems prohibitive from the standpoint of economy, and there is, in addition, the acute danger of fusing the chloride ore salts during heat treatment.

2—Fixing the base exchange ion with a radical of sufficient activity and tenacity to preclude exchange with the amine collector. Though this possibility is still under investigation, its feasibility is hinged on the unit cost of a reagent of this category and possible interference with potash collection by the amine.

3—Blinding the structural interstices of the montmorillonite prior to the addition of the cationic collector. This method of treatment has found application in the mineral dressing industry, and its class of reagents are often referred to as depressants, or auxiliary agents. However, it is felt by the authors that the term *blinding agent* is more descriptive of the action in this particular instance. This scheme of clay control was selected for the Duval process.

In tailoring a blinding agent for the process, numerous characteristics are important, see Fig. 5. The reagent must be extremely compatible with the adsorptive surface of the montmorillonite, hence a hydrophilic or hydroxyl-laden molecule. Molecular structure should be non-anionic to prevent electrostatic interference. Configuration of the reagent must permit easy access between the clay sheets

when they are normally extended by hydration. Individual attractive forces will at best be relatively weak; therefore the molecule should be as large as possible so that a mass bonding effect will be obtained. Additionally, high molecular weight, long-chain molecules, by virtue of their protruding bulk, could temporarily armor the clay particle against penetration by the cationic collector. Hydroxyl configuration should be such that the free groups are *cis*, thus augmenting the weak bonding forces by chelation as suggested to the authors by A. M. Gaudin. From the practical and operational standpoint, the material should be cheap, naturally-occurring, and of dependable quality and quantity.

Fulfillment of these specifications is amply satisfied by the reserve polysaccharide mannogalactan found in the endosperm of several pod-bearing legumes. The actual product in use⁷ is General Mills' *guar*, milled from domestic supply. The configuration is postulated by Whistler and Smith to be a straight chain of D-mannopyranose units joined by β D-1, 4' linkages, with every other unit, on the average, bearing a side chain of one D-galactopyranose residue connected by an α D-1, 6' linkage. This configuration is pictured in Fig. 6. Molecular weight is estimated variously up to 500,000, which would yield a molecule of approximately one micron in length. It is interesting to note that this is larger than the average montmorillonite clay crystal. Again, using Hendricks' figure of 8 million square centimeters of sorptive area per gram of clay, the addition of 1.5 lb of guar per ton of ore feed would give complete initial coverage of all clay surface present in the flotation feed. In as much as one-tenth quantities of guar give the desired result, obviously satisfactory blinding action occurs with the coverage of only the more readily exposed clay surface.

Prior practice entailed the use of starch as the blinding agent. While results with starch are acceptable, they do not equal those obtained with

mannogalactan. Blinding agents are at best of a short-lived nature because of the greater attractive forces in play between the cationic collector and the montmorillonite. In the authors' opinion, the basic superiority of mannogalactan lies in the increased bonding strength which renders it less susceptible to displacement by the collector. In practice, one-quarter quantities of mannogalactan give equal flotation results and a marked improvement in circuit brine clarity. This latter point, as seen in Fig. 7, clearly illustrates the advantage of tailored reagents in closed circuit systems. The general refinery flow diagram in Fig. 8 shows the adaptation of the Krystal unit and the conditioning technique applied to mannogalactan.

In summary, the use of temperature control and tailored reagents combine to give the Duval Sulphur & Potash Co. a highly efficient process, unaffected by seasonal temperature variations.

Acknowledgments

The authors wish to acknowledge the generous assistance of O. A. Moe and W. A. Jordan of General Mills, Inc., in developing mannogalactan as a clay-binding agent. They also express their thanks to Hans Svanoe of the Struthers-Wells Corp. for his valuable consulting service in applying Krystal practice to potassium chloride. In addition, the

classic works of the referenced authors were of immeasurable help in clarifying the role of clay minerals in flotation processes.

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Wet Cleaning at the Tralee Preparation Plant

by Percy Gillie

THE Tralee preparation plant, owned and operated by the Semet-Solvay division of Allied Chemical and Dye Corp., is located on the Virginian railway, near Mullens, Wyoming County, W. Va., and the mine, which is in the Winding Gulf district, produces coal exclusively from Pocahontas No. 3 seam. The property, comprising 4900 acres, is leased from the Pocahontas Land Corp. Prior to 1930 four separate mines were operated in this tract, some having been started about 1916.

Plans for the preparation plant were formulated soon after World War II. Construction began in the spring of 1948, operation early in 1950. The plant was designed to be operated with minimum labor. It was equipped for production of first quality graded

sizes of coal for house heating and $\frac{3}{4}$ x0-in. slack suitable for high-standard metallurgical coke.

In the beginning, a thorough investigation was made of the property by means of core drilling, prospect openings, and sections from shallow mines on the property, as well as sections and analyses from mines in the immediate vicinity. After the extent of desirable coal had been established and the average seam section and nature of overlying and underlying strata had been determined, it was decided to install a system of mining which would load the full seam and would clean the product mechanically.

Pocahontas No. 3 seam is characterized by draw slate and bone and rock partings. It was not known what the washing difficulties and cleaning plant requirements would be for processing the raw product of full mechanical mining. Arrangements were made, therefore, through the courtesy of the property owner, to obtain a large sample from a section of the mine along the adjoining property line. The

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Discussion on this paper, TP 3641F, may be sent (2 copies) to AIME before Jan. 31, 1954. Manuscript, March 1, 1953. Revised July 7, 1953. Joint Session of AIME and West Virginia Coal Mining Institute, White Sulphur Springs, W. Va., November 1952.

place selected was representative of natural conditions observed in faces of entries in old workings of abandoned mines on the Tralee lease. To simulate mechanical loading, since the adjoining mine was operated by hand loading, drill holes were charged with sufficient explosives to pull down draw rock, rock partings, bone, and coal together. Large pieces of draw slate and bone were then broken up with a sledge hammer. A test on this material provided data to be used in designing the cleaning plant.

The washing studies also indicated that full mechanical mining would produce a raw product containing not only a very large amount of high gravity material but also considerable material of intermediate gravity which would increase materially the difficulty of separation. Previous washing tests made on less difficult coals in both full scale and pilot plants, and investigations and sampling in the cleaning plants throughout the eastern coal fields, over a period of several years, indicated that it would be necessary to wet wash both the slack and the 6x $\frac{3}{8}$ -in. coal if the desired cleaning results were to be obtained with a minimum loss of coal in the refuse. As a result of these studies, a Rockmaster roll crusher, a Chance cone, Deister concentrator tables, and Bird filters were selected for the task.



Fig. 1—View of the Deister floor in the Tralee plant.

These six screens are unique in that they make three products, namely, refuse, coarse coal, and slack. When the +6 in. mine product is run through the crusher the draw slate tends to break into slabs. These large slabs are screened out by the top deck and go directly to the refuse bin. The second product is 6x $\frac{3}{8}$ -in. coal which is cleaned in the Chance plant, and the third product is $\frac{3}{8}$ in. slack which is cleaned in the Deister plant. The 6x $\frac{3}{8}$ -in. raw coal is re-screened over a low head screen before it is sluiced into the cone, the purpose being to remove all the fines created by breakage or poor screening. This reduces silt accumulation in the Chance system.

It will be noted from data in Table II that approximately 9 pct of the raw feed with an ash of 74 pct is rejected by the six Ripl-Flo screens with 44 pct of the product going to the Chance cone and the remainder of 47 pct going to the Deister plant, which is shown in Fig. 1.

The coal is cleaned in a 13-ft 6-in. cone with 60 P-2 sand. The washing gravity in the cone is mainly controlled by varying the amount of water recirculated and by the size of sand retained. The amount of water is regulated by valves on each of four lines. Each of these lines supplies water to a ring from which the water is introduced into the cone by nozzles equidistant around the periphery of the cone. The nozzles point down parallel to the side of the cone so that material is flushed by down currents as well as rising currents of sand and water.

Table I. Variations in the Raw Product of Full Mechanical Mining

Draw Slate, 18 In.			Draw Slate, 0 In.		
Type of Coal	In.	Ash	Type of Coal	In.	Ash
Coal	10.5	3.47	Coal	6.5	4.72
Gray coal	1.5	24.71	Gray coal	2.0	9.55
Slate	5.5	69.64	Bone	4.0	57.56
Coal	2.0	10.41	Coal	6.0	8.27
Bone	4.0	45.44	Coal	16.0	2.52
Coal	3.0	7.48	Bone	2.0	21.96
Coal	18.5	4.96			
Bony coal	1.0	14.36			
Coal	9.0	2.98			
Bony coal	1.0	18.76			

The Tralee coal cleaning plant is the first in the Pocahontas field equipped to clean the entire mine product with a Chance cone and Deister concentrator tables. This plant is now processing a raw product from which approximately one ton of material is rejected for each ton of clean coal loaded in the railroad car. This large amount of refuse is rejected entirely by mechanical means, as there is no picking table in the plant.

During the first few months of operation, it was found that long slabs of draw slate, when less than 6 in. thick, would pass through the crusher and become fouled in the cone refuse gates. This was overcome by the addition of scalping screens to remove the larger pieces.

The coal seam varies from 32 to 60 in., depending on the amount of bone and middleman, and is overlaid with a draw rock varying from 0 to 18 in. Typical variations, shown in Table I, produce wide fluctuations in the amount of material which must be rejected by the cleaning plant.

The coal is fed from a 50-ton dump hopper at rate of 450 tons per hr to a 6x10-ft Ripl-Flo screen that removes the -6 in. material. The +6 in. material is crushed in a 30x72-in. Rockmaster crusher, recombined with the throughs, and transported by belt conveyor into the main tippie. This material is distributed by means of a drag conveyor over six open top hoppers which feed to six 6x16-ft Ripl-Flo screens by means of adjustable startgate feeders.

Table II. Run of Mine Feed After Crushing

Product, In.	Wt, Pct	Ash, Pct
+6	8.6	74.77
6 x $\frac{3}{8}$	43.7	45.43
$\frac{3}{8}$ x 0	47.7	14.60

Control by size of sand is effected in two ways, by purchasing specifications and by control of the sand classifier. Sand purchased is of 42 to 63 fineness, American Foundrymen's Association standards. This is the theoretical mesh of the screen with which the average size of a grain of the sand would correspond. The finer sands are more suitable for low gravities, and the coarse sands for high gravities. During operation with a given grade of sand, the average size of the sand kept in circulation in the cone can be varied if water velocity is changed in the classi-

fier column, so that more or less sand is dropped out of the bottom of the vessel. When the sand pump is slowed down, the relative amount of sand in the cone and in other parts of the system can be varied. Return of sand by the pump can be slowed down by injection of fresh water into the pump suction. The operating gravity may change several hundredths because of a build-up or decrease in water solids in the system. It is important to check this build-up when there is a large amount of material near the operating gravity. It has been found necessary to use plenty of sand to maintain the efficiency, sand losses having run as high as five lb per ton during wet seasons. These sand losses can be minimized by wet screening ahead of the cone and other devices, but such screens were not incorporated in the design of the Tralee plant.

When operating at 1.40 sp gr the sinks in the clean coal vary from 0 in the larger sizes to 2 pct in the pea size as shown in Table III.

Table III. Cone Products from Operation at 1.40 Sp Gr

Sp Gr	Wt, Pct	Ash, Pct	Accumulative	
			Wt, Pct	Ash, Pct
6 x 3½-In. Egg				
-1.35	58.7	5.01	58.7	5.01
1.35 to 1.40	41.3	8.10	100.0	6.29
+1.40	0			
3½ x 1¾-In. Stove				
-1.35	66.3	4.74	66.3	4.74
1.35 to 1.40	33.0	8.50	99.3	5.99
+1.40	0.7	14.41	100.0	6.05
1¾ x ¾-In. Furnace				
-1.35	70.7	4.63	70.7	4.63
1.35 to 1.40	28.3	8.52	99.0	5.74
+1.40	1.0	13.27	100.0	5.82
¾ x ¾-In. Pea				
-1.35	77.2	4.25	77.2	4.25
1.35 to 1.40	20.8	8.24	98.0	5.10
+1.40	2.0	12.51	100.0	5.25

This cone is the open discharge type which means that the refuse falls directly from the cone onto a desanding and dewatering screen without the necessity of using a bucket elevator. When operating at 1.40, Table IV, the float in the refuse is 7/10 of 1 pct and the ash content of the refuse is 65.2 pct.

Table IV. Cone Refuse Float and Sink Analysis 4-Hr Sample During Operation at 1.40 Sp Gr

Sp Gr	Wt, Pct	Ash, Pct	Accumulative	
			Wt, Pct	Ash, Pct
-1.35	0.1	5.15	0.1	5.15
1.35 to 1.40	0.6	11.50	0.7	10.60
1.40 to 1.45	2.2	15.94	2.9	14.65
1.45 to 1.50	5.0	20.50	7.9	18.35
1.50 to 1.60	6.3	26.50	14.2	21.96
1.60 to 1.70	5.1	37.70	19.3	26.12
+1.70	80.7	74.55	100.0	65.20
Total	100.0	65.20		

The clean coal which floats in the sand and water mixture is discharged onto a Parrish-type shaking screen, the first section of which is used entirely for dewatering and desanding. The second section is used to size the coal into four grades: +3½ in. egg, 3½x1¾-in. stove, 1¾x¾-in. furnace, and ¾x¾-in. pea. These grades are loaded by means of drag-type loading booms equipped with oil sprays, oil metering devices, and degradation screens. The degradation is conveyed back and in turn is rescreened over a double-deck vibrating screen which dewateres and sizes the pea coal. This screen is equipped on the

bottom deck with a ½-mm screen cloth. The plus material goes to the slack car and the minus material, which is mostly sand and silt, flows to the sand sump. The screening plant is equipped with a mixing conveyor so that any of the sizes may be blended together, loaded into the slack car, or conveyed to the house coal bin.

The ¾x0-in. slack which is screened out by the six Ripl-Flo raw coal screens is elevated by means of a drag conveyor to the top of a 100-ton storage and surge bin at a maximum rate of 250 tons per hr. One of the most important functions of this bin is to provide sufficient storage so that the table plant can be operated continuously. The coal fed from this bin is regulated by a variable speed rotary feeder which discharges it into a flight conveyor that elevates the coal to what is called a splitter box. At this point, the dry slack falls into the rising current of water which is used to wet the coal and convey it to the table distributors. The quantity of water used in the splitter box amounts to about 1400 gal per min. The purpose of the distributor is to divide the feed equally and without segregation to each of the tables.

The coal and water are fed to the concentrating tables at the rate of 10 tons of coal per table per hr. To clean the quantity of fines, it was necessary to have 20 tables, which were installed on two floors. Feed requirements are shown in Table V.

Table V. Screen Analysis of Raw Slack to Concentrating Tables

Size, Mesh	Wt	Ash
+ ¾	3.1	49.72
- ¾ + ¼	12.3	19.07
- ¼ + 8	24.0	13.73
- 8 + 16	19.8	12.24
- 16 + 30	15.5	11.29
- 30 + 50	10.8	10.24
- 50 + 100	6.3	11.56
- 100 + 200	3.4	11.12
- 200	4.8	21.53
Total	100.0	

The tables are the diagonal deck type. Approximately equal weights of coal and water are fed at one corner of the table. Additional dressing water is added at two points along the back of the table to wash the coal over the riffles, which are diagonal to flow of coal. A suitable conveying motion is imparted to the tables by a 3-hp motor. This motion is parallel to the riffles, shaking the material away from the coal overflow lip at a 45° angle toward the refuse side of the table.

Control of the separation on tables is effected by the following adjustments in order of their importance: 1—end elevation of table, 2—amount of dressing water added, 3—side tilt of table, 4—amount of water added with feed, and 5—length of stroke and number per minute.

Even after seeking the optimum combination of these adjustments it was found that, although the overall ash was satisfactory, there was still an objectionable quantity of intermediate gravity material reporting to the clean coal. It will be seen in Fig. 2 that this high ash material is concentrated in the middling zone of the table. When this high ash material is diverted to refuse, the fine coal also goes to refuse. Coarse bone and fine coal tend to come off the table at the same place, as is usual on tables with unclassified feeds.

It was found that it was very simple to correct this defect by installing a fence along the coal edge

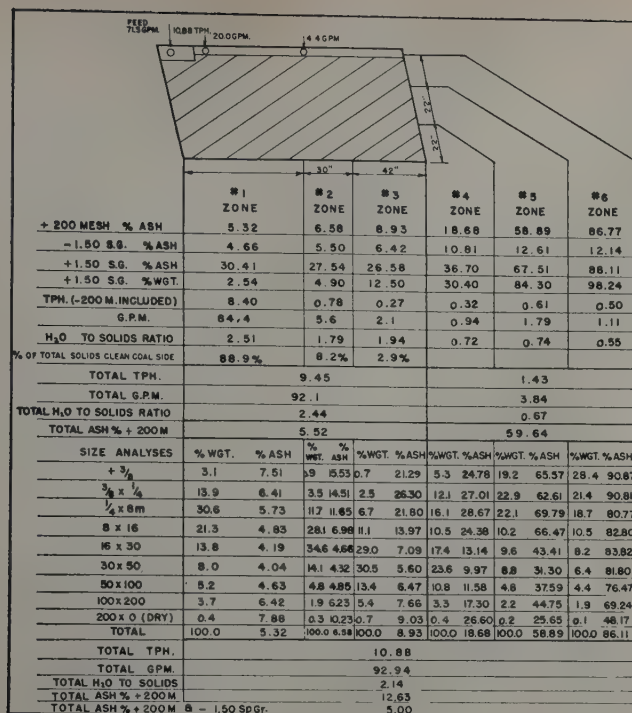


Fig. 2—Zone test on tables (5).

of the table. The fence consists of a 42x4-in. piece of 1/4x3-in. Tyrod screen cloth cut and mounted along the No. 3 zone (Fig. 2) edge of the table with the slots vertical. It works because the conveying action moves large particles away from the fence so that it does not become obstructed. Table VI shows an ash and screen analysis of the table product after installation of the fence.

The clean coal and water from the tables is conveyed by sluices to a 5-way distributor which in turn is used to feed the five 54x70 solid bowl centrifugal dryers manufactured by the Bird Machine Co. The ratio of the water to the coal in the feed is generally 2 to 1 by weight. The Bird centrifugals dewater the coal from a feed moisture of approximately 67 pct to a discharge surface moisture of 6 to 8 pct. The coal from the Birds is discharged onto a flight conveyor which conveys and elevates it to a mixing conveyor or cross belt for loading into railroad cars.

Table VI. Clean Coal* Screen Analysis after Installation of 1/4-in. Fence

Size, Mesh	Wt, Pct	Ash, Pct	Accumulative	
			Wt, Pct	Ash, Pct
+ 3/8	2.2	6.47	2.2	6.47
- 3/8 + 1/2	11.4	5.47	13.6	5.63
- 1/2 + 3/4	24.8	5.16	38.4	5.33
- 3/4 + 1	20.9	4.20	59.3	4.93
- 1 + 1 1/4	10.0	3.85	69.3	4.57
- 1 1/4 + 2	7.2	4.65	76.5	4.57
- 2 + 2 1/2	6.4	6.23	82.9	4.68
- 2 1/2 + 3	2.9	7.51	85.8	4.77

* Washed on 200-mesh sieve before analysis.
Wet -200 mesh discarded.

Also of interest in the fine coal cleaning plant is the installation of a cyclone in the dressing water lines going to the various tables. This removes all the tramp material including the coarse coal from these lines so that the valves do not become clogged. Before installation of this cyclone, constant knocking on the valves was necessary to keep them un-

stopped. The Bird machines as originally furnished operated at a speed of 640 rpm. Tests made at another plant indicated that they could be operated at a slower speed. The Birds at Tralee are now operated at a speed of 480 rpm. This reduces the horsepower requirements of these machines 30 pct and also decreases the amount of breakage and lowers the moisture in the discharge cake. Results accomplished by these changes are shown in Table VII.

The screws in the Bird machine have been redesigned so that part of the screw is used as a centrifugal pump, pumping water out at a high velocity into the partially dewatered coal being conveyed up the so-called beach of the machine. The purpose of this high velocity stream of water is to remove the clay or deslime the slack. The clay and slime retain several times their weight in moisture, so that this desliming has actually produced two results: first, a lowering of the moisture content of the coal, and second, decreasing of the ash of the coal coming from the machine as shown in Table VII.

Refuse from the various tables is conveyed by sluices to a rectangular sludge tank from which the material is removed by a drag conveyor. The material as removed contains approximately 20 pct moisture. The entire amount of refuse from the mine, both mine rock and washer reject, is con-

Table VII. Tests on Bird Machines Operating at 640 rpm vs 480 rpm

Item	Bird No. 1	Bird No. 2
Bird cake, pct total moisture	17.00	10.60
Effluents, pct solids	5.55	8.45
Wash, gpm	0	175
		Bird effluent
Pool, in.	1 1/4	1 1/4
Speed, rpm	640	480
Foxboro reading	310	250
Amperes, input	200	135
Bird Cake Screen Analyses, Wet Washed on 200-Mesh Sieve		
Screen Size, Mesh	Cake, Wt Pct	Cake, Wt Pct
+ 3/8	0.2	0.4
- 3/8 + 1/2	3.7	5.4
- 1/2 + 3/4	17.4	26.6
- 3/4 + 1	17.1	22.4
- 1 + 1 1/4	15.1	13.9
- 1 1/4 + 2	12.3	10.0
- 2 + 2 1/2	10.4	7.6
- 2 1/2 + 3	7.4	3.8
- 3 + 3 1/2	16.4	9.9
Total	100.0	100.0
Effluent Screen Analyses		
Screen Size, Mesh	Wt, Pct	Ash, Pct
+ 100	0.6	10.89
- 100 + 200	1.6	4.93
- 200 + 325	22.3	3.84
- 325	75.5	37.99
Total	100.0	29.68

veyed by means of a belt conveyor to a 240-ton aerial tramway bin. It is loaded from the bottom of this bin into a continuous aerial tramway which conveys and elevates it to a transfer bin at the top of a mountain. It is finally disposed of by a 2-bucket jig-back disposal tramway. The disposal area is sufficient to hold approximately 20 million tons of refuse.

A dumper, cone operator, table operator, pump attendant, and clean-up man are required to operate the tippie and cleaning plant.

No greasers are needed, as all the machinery is lubricated by centralized automatic facilities. There are three men to clean and handle the railroad cars to and from the tippie and one operator for the aerial tramway.

At the present time information received from consumers indicates that an acceptable product is being produced with a minimum of operating labor from a coal seam with exceptionally difficult washing characteristics.

The Status of Testing Strength of Rocks

by Rudolph G. Wuerker

The progress made in testing the strength of rocks and minerals as they are encountered in mine operation is reviewed. An attempt is made to correlate these physical measurements with abrasive hardness, grindability, and behavior in comminution on one hand and fracture of rocks in pillars and roof control on the other.

THIS paper reviews the progress made in testing the strength of rocks, ores, coal, salts, and other minerals as they are encountered in mine operations. It attempts to correlate the results of these physical measurements with technological properties more useful to the mining engineer: abrasive hardness, grindability, and behavior in comminution on one hand, and roof control, fracture of rocks in pillars, and mining methods with controlled caving on the other. In the following pages, the materials discussed will be referred to as *rocks*.

Basic to rock mechanics and comminution are the problems of strength, elastic behavior, and failure, common to all brittle materials. A distinction will be drawn as to theoretical and applied research, and discussion of the progress made in each field will include test data obtained by the U.S. Bureau of Standards,¹⁻⁴ the U.S. Bureau of Mines,⁵⁻⁷ the Iowa Engineering Experiment Station,^{8,9} the Committee on Geophysical Research at Harvard University,¹⁰ Basic Industries Research of the Allis-Chalmers Manufacturing Co.,^{11,12} by Philipps,¹³ and by Mueller,¹⁴ to name only a few.

With refinements of testing methods and increased standardization, more useful and more comparable results have been achieved. This is especially important in testing a material like rock, as the inherent heterogeneity demands careful and exacting procedures. New measuring procedures that appear to supersede well known standard methods have contributed to faster and less costly testing yet have introduced new concepts, with implications as to comparability of results which must be watched. Reference is made to the sonic method for determining elastic properties,⁵ to be discussed in detail below.

Basic Investigations

Historically, all work in the field has started with the simplest determinations such as those for crushing strength, abrasive hardness, and grindability. These serve the limited objectives in the researcher's field of specialization: building construction, road ballast, roof control in mines, comminution, and seismic prospecting. Occasionally, fundamental properties like the modulus of elasticity E and Poisson's ratio ν have been determined with the idea that they might have some bearing on the technological properties of the material under investigation. But it was not until the work of Philipps,¹³ of Harvard University,¹⁰ and of the U.S.

Bureau of Mines^{6,7} that sufficient basic data were collected to allow researchers to go beyond the technological test and find the fundamental laws behind the behavior of rocks in mine and mill operations.

The properties to be looked for are those that describe the elastic behavior of any material, the modulus of elasticity E and Poisson's ratio ν being the ones determinable with least difficulties. Only two such properties are required to compute any other property such as the shear modulus, the modulus of rigidity, and the bulk modulus, all of which are related to each other according to well known equations of the theory of elasticity.¹⁰ In spite of their heterogeneous character, all rocks tested have possessed elastic properties. This does not mean that rocks of the same type always have the same modulus of elasticity, which varies exactly as the crushing strength or any other physical property of a rock can spread over a wide range. This has been explained by imperfections of the material always found in rocks, but to some extent this scattering of data is caused by inaccuracies inherent in the testing methods.

Modulus of Resilience, a Criterion of Failure

Increased availability of E values should allow us to test the validity of the quantity of strain energy theory which has been used in the solution of roof control problems by Philipps¹³ and by Holland.¹⁵ Recently Bond and Wang¹⁹ have applied this theory to explain the failure of an elastic material in comminution. Actually it is a very old theory, proposed as far back as 1885 by Beltrami.¹⁶ By its assumption the condition of yielding is determined by the term

$$M_r = \frac{S^2}{2E} \times \text{volume. Here } M_r \text{ is the modulus of}$$

resilience, and its dimension is inch-pounds per cubic inch, that is, work per unit volume. Its numerical value is equal to the area under the stress-strain diagram. In the foregoing equation S is the yield stress (in psi) in tension or compression, whatever the case may be. E , the modulus of elasticity, is in psi.

The great appeal of Beltrami's concept of stored energy lies in the fact that the two properties which seem to influence failure most, strength and elasticity, occur in the formula for the modulus of resilience. As an illustration of this, the moduli of resilience in compression of some typical materials tested by the U.S. Bureau of Mines⁶ have been plotted in Fig. 1. The sample of concrete of conventional mix is shown only for the sake of comparison. Its determination was made in the Department of Mining and Metallurgical Engineering, University of Illinois. The values of the moduli of resilience of the various specimens in the plot are:

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Discussion on this paper, TP 3556A, may be sent (2 copies) to AIME before Jan. 31, 1954. Manuscript, Feb. 18, 1952. New York Meeting, February 1952.

Rock		In.-Lb Per Cu In.	
Jaspilite	(8.4)	=	448.0
Hematite ore	(8.3)	=	134.0
Amphibolite	(17.2A)	=	124.5
Marble	(1.1)	=	66.3
Concrete		=	5.0
Sandstone	(1.4)	=	62.5

The concept of strain-energy, expressed by M , and represented graphically by the area under the stress-strain diagram, shows clearly the dependence of our rocks on the two variables, strength and elasticity. Although in the plot the marble (1.1) and the sandstone (1.4) have about the same modulus of resilience, it is evident from their stress-strain diagrams, shown in Fig. 1, that the former sustains a much higher load and a smaller deformation, and that the latter sustains a smaller load but offers a much greater yield. The overall energy, nevertheless, to crush 1 cu in. of either material is, theoretically, about the same. Whether or not this approach will lead to better understanding of the problems of comminution and rock mechanics will depend upon further data. In most cases, insufficient elastic values are given in published data on crushing and grinding.

Besides the need for a greater number of E values, there is also a need for agreement on the methods of their determination and on interpretation of test data. For only in exceptional cases in the static determination of the modulus of elasticity of rocks are straight-line relations between stress and strain obtained.

In the ASTM standard test for determining compressive strength of rocks (C 170-41 T), modified somewhat by the U.S. Bureau of Mines,⁶ three types of stress-strain diagrams are usually obtained: straight line, S-shaped, and steady-curving.

Fig. 2, which is taken from Kessler's¹ tests, shows repetition tests with six specimens of limestone. The parallelism of the curves upon repeated loading and unloading is indeed remarkable. While it is typical for dense, homogeneous rock, there are not such straight and reproducible lines for coarser-grained and less consolidated rock, especially upon repeated loading. But the coarser rock always shows an elastic phase, as can be seen from the stress-strain diagram of coal loaded to fracture, Fig. 3, which is typical for an S-shaped curve. This type of stress-strain curve is the most common, and has been obtained from rocks, concrete, ore, salt, wood, and similar more or less coarse-grained and brittle materials. Three different phases can be distinguished here, 1—a slope from the point of origin to 2—a straight line portion, and 3—a bending away from the straight line toward the point of failure. Phases 1 and 3 are often more or less pronounced, although sometimes they are completely missing.

Phase 1, curving up and showing increasing values of the modulus of elasticity with increasing stress, is probably due to the increasing compacting of the specimen and to an adjustment of the specimen in the fixtures. This phenomenon has been given undue attention by many investigators, especially the geophysicists, where rocks and similar materials were not loaded to failure. This is understandable when repetition tests are made, but it is not a recommended procedure for obtaining full knowledge of the way rocks behave up to, and at,

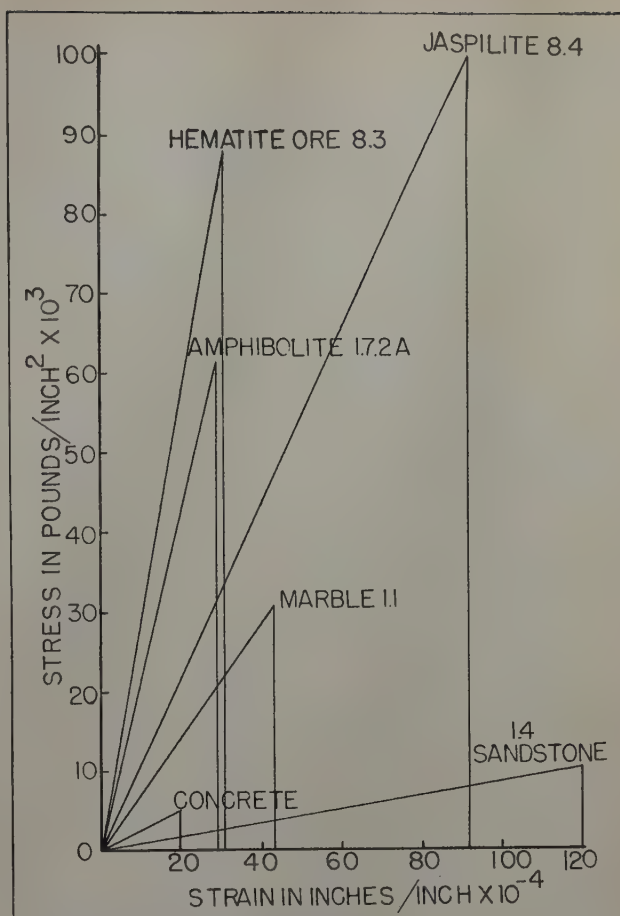


Fig. 1—Moduli of resilience of some typical rocks tested by the U. S. Bureau of Mines. The dolomitic marble is found at Cockeysville, Md. The sandstone, which is porous and weakly cemented, is from Amherst, Ohio. The intricately banded, medium-to-coarse-grained hematite is mined at the Soudan iron mine, St. Louis County, Minn., by the Oliver Mining Co. The jaspilite, comprised of alternating bands of chalcedony, cherty hematite, hematite, and magnetite is also from this mine. The amphibolite, two-thirds of which is composed of actinolite, also contains quartz, feldspar, epidote, and sphene. The samples of amphibolite were taken from the Nundydroog Gold mine, Nundydroog Mines, Ltd., at Oorgaum, Mysore State, South India.

failure. If a test is run until fracture, there is always a portion in the stress-strain diagram, in most cases having the steepest slope, through which a straight line can be drawn. This might be called the *apparent* modulus of elasticity, analogous to the *apparent* specific gravity in the ASTM tests, although this term is not in common use.

Another stress-strain curve of a brittle material of the steadily curving type, phase 3, is given in Fig. 4. It is taken from a concrete compression test, and like Fig. 3 it shows a steady curving of the line toward the point of fracture. As it is difficult in this case to find a straight-line portion, or even the *apparent* modulus, concrete manufacturers, who have the most experience with such a curve, work with the following moduli of elasticity that can be represented by a straight line:

1—The initial tangent modulus. This is the line drawn as a tangent to the stress-strain curve through the point of origin, a in Fig. 4,

2—the tangent modulus for a given stress, b in Fig. 4, and

3—the secant modulus at a given stress, usually the proportional limit stress, if this point can be

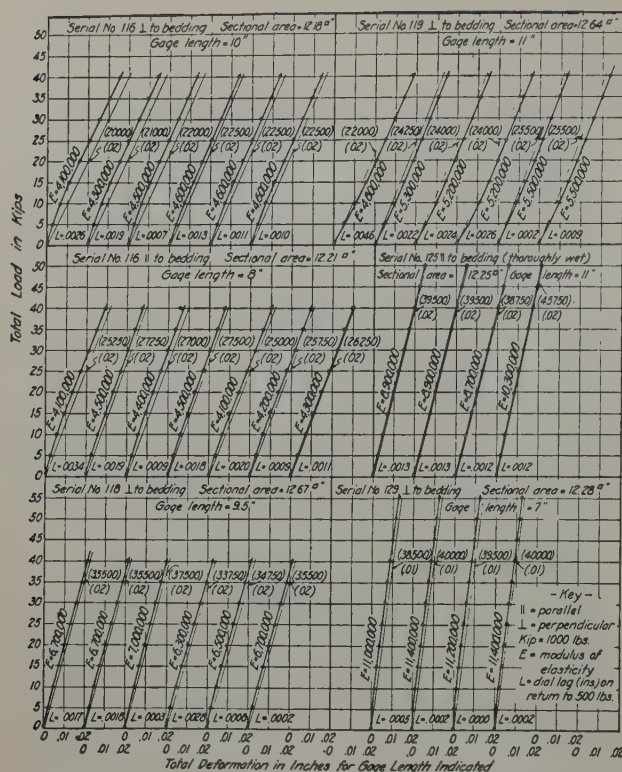


Fig. 2—Elastic deformation curves obtained for specimens of limestone, showing repetition tests. (After D. W. Kessler and W. H. Sligh, U. S. Bur. of Standards TP 349).

found on the curve, or at any other assumed stress, c in Fig. 4. This secant modulus of elasticity is the one most commonly used in design. As it is always less than the actual modulus of elasticity and less than the tangent moduli, it is also the safer one.

If an attempt is made to draw the secant modulus to a curve as in Fig. 3, which is a more common type of curve than the one in Fig. 4, the difficulty arises as to how to fit in a secant line which should go through the point of origin. This is why the *apparent* modulus of elasticity has been suggested.

Until the advent of the sonic testing method for determination of elastic properties, only static tests were used for measuring of E and ν . The modulus of elasticity thus found would be any of the three aforementioned tangent moduli for a given stress (a and b in Fig. 4), or any of the secant moduli (c) that could possibly be fitted into the stress-strain diagram, or the *apparent* modulus of elasticity. E determined by the sonic method is always the initial tangent modulus (a in Fig. 4) and is always a higher value than the moduli gained from static testing. There is obviously the danger of an inaccuracy in the indiscriminate use of existing testing methods, a practice which should be avoided if the experimenter wishes to arrive at definite, indisputable physical data.

E of brittle materials is not a constant property but changes with the speed of loading. Philipps, see Fig. 5, has published stress-strain diagrams of shales tested at various speeds. E from a test that reached 12,000 psi in 10 min was 25 pct less than that at immediate deformation. The rate of loading of 100 psi per sec, as specified in ASTM and U.S. Bureau of Mines procedures, would be represented by the 2-min curve in Fig. 5 which gives a value of about 20 pct less than that obtained by a rapidly applied load, as in the sonic test. Similar discrepancies can be expected with indiscriminately used

values obtained by different methods or by any of the three possible E 's from the static method.

To summarize this point, a decision by a standardizing agency on this question would greatly contribute to usefulness and reproducibility of future data. Such an agreement would be as helpful as the specification of a speed of loading 100 psi per sec in the standard compression test. Until then, elastic properties of rocks determined in the past by various methods should be used judiciously.

Applied Investigations: Relation Between Basic and Technological Properties

The fundamental physical properties discussed in the foregoing will in all probability be used only in the study of the problem of fracture of rocks. Their determination requires testing equipment that will be found only in universities and testing laboratories. Another disadvantage is that most tests (except the sonic) are destructive and require a greater number of specimens, leading to increased costs. Furthermore, a statistical average of a great number of tests has to be taken, as rocks do not possess definite properties. This is explicitly specified in the U.S. Bureau of Mines standardized tests.⁶ Here, in the case of compressive strength, it is recommended that the average of 10 determinations be used.

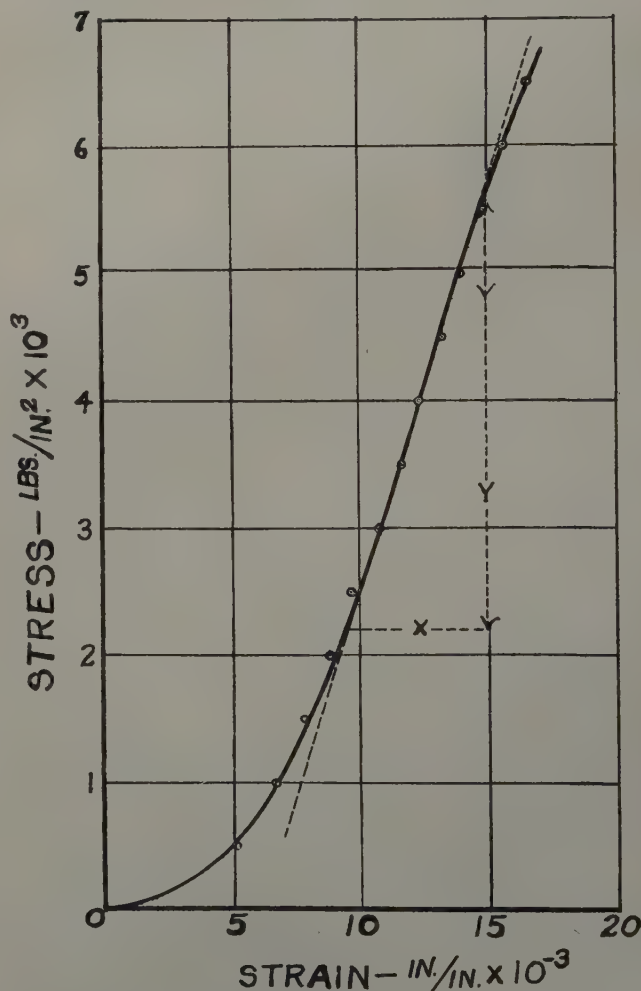


Fig. 3—The stress-strain curve of a coal sample in compression. Specimen IC₃ 1½x1 9/16x2 1/16 in. April 29, 1952.

Olsen machine. Modulus of elasticity = $\frac{Y}{X} = \frac{3400}{0.0055} = 618,000$ psi.

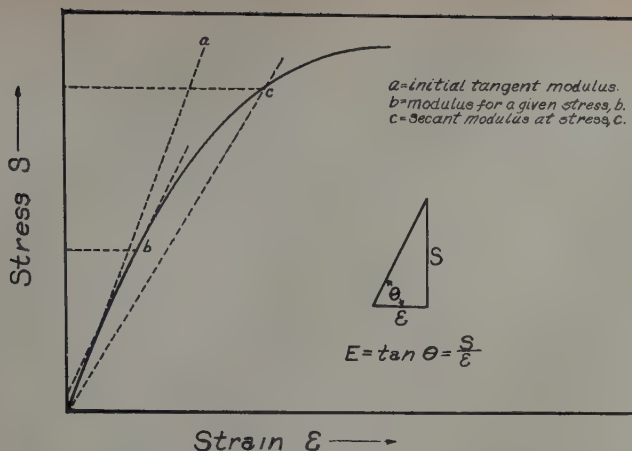


Fig. 4—The stress-strain curve of a brittle material. Various moduli of elasticity.

The trend will be necessarily toward non-destructive and less expensive correlative tests, just as in metal shop work elastic properties, strength, and workability of a piece can be determined by such a simple test as, for example, the Rockwell hardness test.

Correlation of Strength Properties with Density and Absorption

Most of the researchers quoted here have given correlations of elastic and strength properties of rocks with easily determined quantities like density, absorption percentage, or hardness. Correlation with density would be an ideal and most simple procedure, but in all the literature reviewed only one successful case has come to the writer's attention. Kessler-Sligh¹ in their investigation of the physical properties of limestones arrived at a straight-line expression for the ratio of compressive strength to density which is very impressive but which could not be found with any other rock. Limestone apparently is quite free of cavities, bedding planes, and other imperfections, and will give a fair straight-line expression. It might be possible that single rock types of a given mine or a given locality may give similar simple relations. But applied to groups of rocks, like the 97 rocks tested by Griffith, comprising igneous, metamorphic, and sedimentary types, or applied to the 100 or more rock varieties from operating mines or mineral-investigation projects tested by the U.S. Bureau of Mines, this procedure is a failure.

Griffith⁸ has chosen a property closely related to density, namely, absorption. His argument is that "the physical behavior of rocks is conditioned much more by their relative degrees of molecular dispersion or states of aggregation than by their chemical constitution." His argument is very plausible in the extreme case of diatomaceous silica and chert. He says, "Both are modes of silica, but the one absorbed 154.6 percent of water by weight and supported 477 psi., while the other absorbed hardly any water and carried 86,300 psi." A similar test made at the Department of Mining and Metallurgical Engineering at the University of Illinois with coal and coke gave for coal 4000 psi compressive strength and 16 pct absorption and for coke 780 psi supported unit load at 59.8 pct absorption (averages of three tests). While for such extreme cases Griffith's argument might be acceptable, there is in his plot of compressive strength vs absorption a crowding of values in the absorption range from 0 to 2 pct, and the parameters given by

him become meaningless here. The majority of his measured values fall in this range. Salt rock, coal, and many sedimentary rocks will not be amenable to the absorption test at all, as they disintegrate during the boiling period required in this test.

Correlation of Strength Properties with Rebound Hardness

Better results are gained by the correlation of compressive strength with rebound hardness as determined with the Shore scleroscope. This too has been suggested by Griffith, and the value

$$S_c = 300 h \left(1 \pm \frac{1}{10} \right)$$

has been given by him for the compressive strength (S_c) as a function of scleroscope hardness h for averages of the data rocks tested at the Iowa State College Engineering Experiment Station.⁸

This correlation applied by the author to the group of more than 100 rocks studied by the U.S. Bureau of Mines gave better results than any other attempt. As can be seen from the curves on Fig. 6, the relation of compressive strength to scleroscope hardness can be fairly described by $S_c = 400 h_{scler.}$ as average value. Griffith's formula $S_c = 300 h_{scler.}$ forms, with the rocks of the U.S. Bureau of Mines test series, the lower limiting value and $S_c = 500 h_{scler.}$ the upper parameter, although the few extremely strong rocks found by the U.S. Bureau of Mines lie even beyond this formula. Some of the high strength values are not shown at all, as the ordinate does not extend beyond 50,000 psi. The correlation should be considered as a first attempt. Possibly a parabola would fit the plotted values better than a straight line, and more definite values could perhaps be obtained by correlating the properties of single rocks or rock types, as in the case of the limestone shown in Fig. 2, instead of a group composed of 100 and more different rocks.

The gratifying results with this correlation attempt encouraged the author to test relations of scleroscope hardness with other technological properties. In Fig. 7 scleroscope hardness is plotted against abrasion hardness, as determined by the modified Dorry abrasive hardness test.⁹ The values are again from the U.S. Bureau of Mines test series⁹ of more than 100 different rocks of all sorts and of some iron ores. In spite of their diversified origin, composition, and texture, parabolas could be found

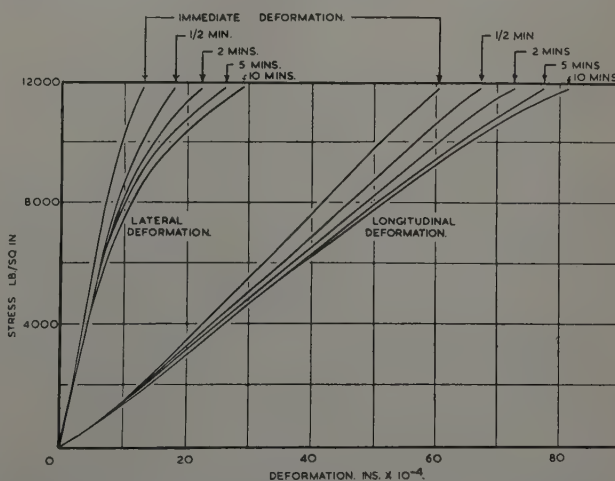


Fig. 5—Effect of time of loading on stress-strain diagram, compression. (After D. W. Philipps, *Colliery Engineering*, August 1948, p. 281.)

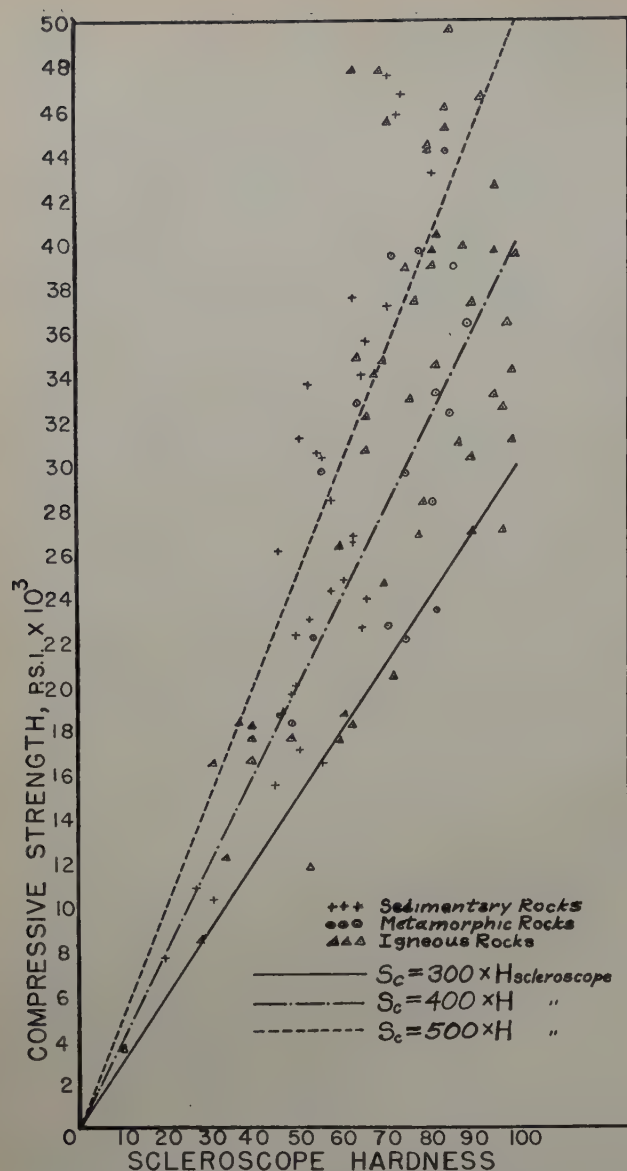


Fig. 6—Scleroscope hardness vs compressive strength of rocks tested by the U. S. Bureau of Mines. (Data from U. S. Bur. Mines R. I. 4459.)

to describe the parameters or an average value for this relation. Here again it is felt that a more definite correlation might be obtainable if rocks of only the same type were tested and correlated, instead of groups of rocks.

There are no more test series available in which scleroscope hardness values have been obtained, so that no further proofs for the validity of the relations established in the foregoing can be obtained. The scleroscope hardness test is fast, inexpensive, and non-destructive. If its general applicability could be established, a way would be found to get strength or other physical data without having need for the apparatus of a testing laboratory. Testing, by means of correlation tests, should thus be possible at mine laboratories, whenever problems of roof control, drill speed progress, and the like arise.

Hardness Correlated to Complex Properties Used in Comminution

This interesting possibility of correlating hardness with those properties of rock that are extremely difficult to determine, such as surface-

energy relationships in comminution, has been intimated by Schellinger¹⁷ and by Kwong-Adams-Johnson-Piret.¹⁸ Although only a few rocks were tested by these authors, certain agreement with Moh's hardness was noted. Schellinger states that if a different hardness number than the non-linear Moh's scale had been used, a straight-line relation might have been possible, see Fig. 8.

Conclusions

1—Experimental work has proved the applicability of the theory of elasticity and of the standard methods of testing materials to rocks, ores, and other minerals with which mine and mill operators have to deal. This statement holds unrestricted for igneous and metamorphic rocks and so-called *strong* ores. Sedimentary rocks are sometimes not very consistent in their elastic behavior and must be evaluated with special statistic procedures. They are in many respects similar to common engineering materials like cement or wood. The geophysicist in seismic work likewise utilizes the elastic properties of sedimentary rocks in computing the travel speed of a wave, and the petroleum engineer, too, in reservoir engineering, also works with physical properties that are indeterminate. In neither field of activity has the success of the methods been influenced. In soil mechanics much less consolidated material than the sedimentary rocks of mining areas has been evaluated as to its elastic and plastic properties and utilized successfully in design.

2—Knowledge of strength properties of rocks as used in fundamental studies would be increased by more exact testing and by agreement on procedures. More definite data could without doubt be obtained if all pertinent tests were made with selected rocks that are typical for their strength properties. Still better, man-made rocks fabricated under controlled condi-

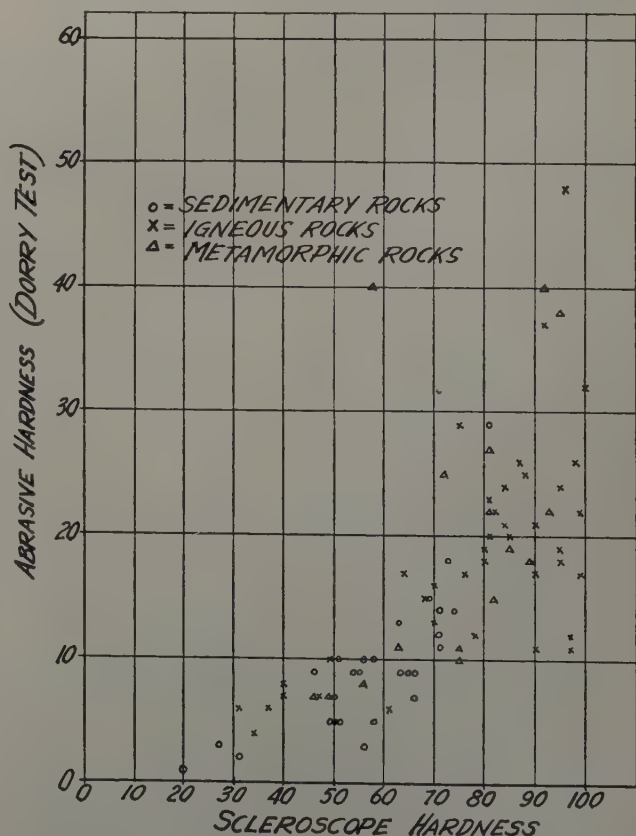


Fig. 7—Scleroscope hardness vs abrasion hardness of rocks tested by the U. S. Bureau of Mines.

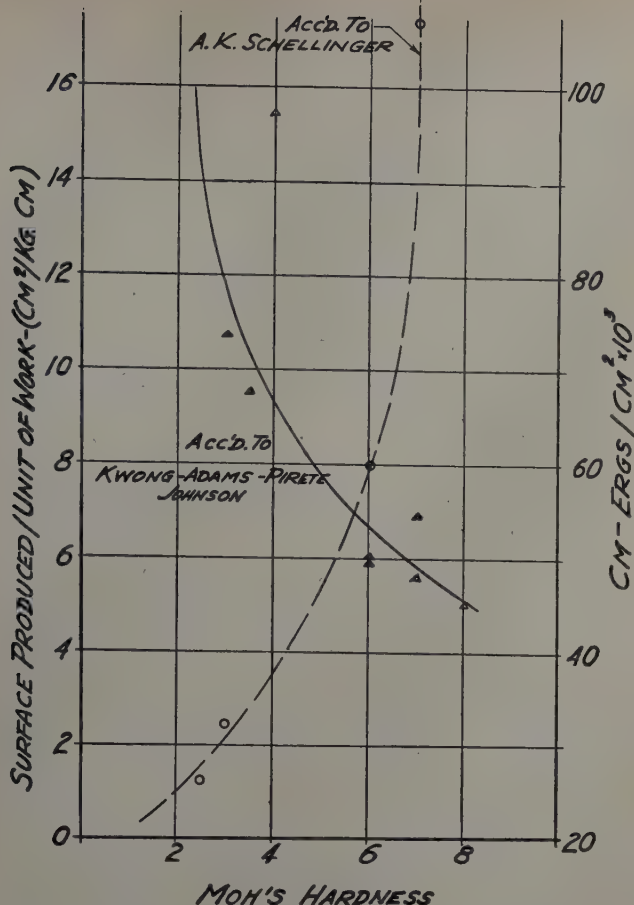


Fig. 8—Moh's hardness related to comminution quantities.

tions could be employed for making the necessary number of repetition tests. For low strength tests neat cement or cement mortar could be used. Igneous rocks could be simulated by bricks ranging from conventional clay bricks up to silica fire-bricks. Thus the inherent heterogeneity of natural rocks and the resultant diverse strength properties could be compensated for and the number of variables reduced.

3—Rocks of importance only to a special mine or to a localized mining district preferably should be tested at the site of operation. Correlations with technological properties or with the costs of various mining processes are of interest only to the particular case. However, methods and testing equipment should be the same as those used in the search for basic physical properties.

Harley¹⁰ in 1926 proposed a system of ground classification for estimating the collective strength of rock as it would affect stoping operations, the drilling rate for setting contract work, and other operational data. Although some suggested testing methods reflect the status of testing of a generation ago, his paper states, though only qualitatively, the same facts that have been proved quantitatively by the later more exacting test which served as basis for this review.

Acknowledgments

This work is part of a comprehensive program of investigating strength properties of rocks undertaken by the Department of Theoretical and Applied Mechanics at the University of Illinois. The wholehearted support given the project by Professor H. L. Walker, Chairman of the Department, is herewith gratefully acknowledged. The tests performed

at the University were made in the Department of Applied Mechanics. Its staff extended all possible assistance. Samples for the tests at the University of Illinois were supplied by the Murdock Mine of the Bell and Zoller Coal and Mining Co., Murdock, Ill. Most of the actual testing, computing, plotting, and preparation of data for the manuscript was done over a period of almost two years by Messrs. J. R. Miller, N. Szabo, A. Risi, and J. W. Snarr, all former students of the Department of Mining and Metallurgical Engineering at the University of Illinois.

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The Mineralogy of Blast Furnace Sinter

by Hobart M. Kraner

THE mineralogy of blast furnace sinter is of interest because its mineral content is one of the important factors contributing to its character. There are so many other factors affecting the properties of the sinter, however, that it is well to mention them here. The proportion and character of the raw materials, that is, raw ores, concentrates, returns, and fuel, as well as the mixing and the water content, all have a marked effect on the physical properties of the product and the degree to which sintering action can be carried on. The process of sintering is a relatively fast operation. In as much as appreciable time is required to carry on processes of fusion in such masses of low thermal conductivity, large lumps of hematite ore frequently remain unfused and partly unchanged in state of oxidation in the sintering process. The kind, the grain size, and the amount of fuel used affect both the completeness of the fluxing reaction and the prevailing atmosphere. The rate of reduction in laboratory tests is not only dependent upon the state of oxidation of the sinter but also upon the sizing and porosity.

Atmosphere and temperature affect the state of oxidation of the iron oxide, and the atmosphere alone may determine the ferrous minerals that finally develop. The rate and extent of cooling, the type of coolant, the subsequent handling, and screening all have serious effects upon the type of sinter that eventually enters blast furnace bins.

The degree to which actual fusion or fluxing takes place in the sintering operation has a marked effect upon density. A sinter which has been extensively fused by high content of fuel in the batch will no doubt have a higher weight on the bulk basis than one which had a lower fuel content. As high temperatures are required to do this job, the iron oxide under these conditions will be largely magnetite. Sintering at low temperatures to produce larger proportions of hematite means a decrease in the amount of liquid formed and a much more sensitive bonding process. In this case the liquid must be distributed more uniformly and thereby used more efficiently than would be the case where higher temperatures were permitted to prevail more or less indiscriminately.

Where coarse ore particles are used in a sinter mix it is not expected that any particles coarser than $\frac{1}{4}$ -in. can be fused and incorporated in the system to such an extent that the gangue contained within these lumps will have been converted or fused by the sintering process. It is for this reason that coarse ore, returns, or both, in a sinter usually result in a sinter which breaks easily and at the same time may contain some of the original minerals of the lump, such as quartz and hematite.

In examination of sinters at Bethlehem Steel Co. minerals such as quartz and corundum have been found, none of which are considered normal associ-

ates of wustite or magnetite. Some degree of heterogeneity or lack of equilibrium is not unusual in the sintering process.

The differences in specific gravity between hematite and magnetite might be ample reason for poor strength in a not very well sintered mass containing coarse particles of ore or returns. The shrinkage taking place in a lump of hematite in its conversion to magnetite by temperature and/or atmosphere is appreciable.

Sintering of ores as it is carried out is crude chemistry, for the grain size is relatively coarse, the application of heat is certainly not uniform, and the time factor is inadequate for other than partial completion of reactions. Coarse lumps of coke or coal cause local heating around these centers, and fuel which is too fine may result in such slow burning that sufficiently high temperatures are not always obtained. High temperatures are essential to the work required. The Swedish practice of sintering is established on the basis of producing an easily reducible product high in hematite. This is achieved through uniformity of grain size in the sinter mix and close control of the temperature through careful regulation of fuel and sintering rates. This produces a sinter which is very tough in character and which has a high degree of porosity. Although the hematite content is not produced upon cooling by drawing air through the mass, there would be greater possibility of accomplishing this reaction with this type of sinter than is the case in American practice. In the latter, the temperatures are so high that temperature alone converts most of the mass to magnetite. The grains are so coarse in the final product that together with the fluxed condition it would be difficult to reoxidize them to hematite upon cooling.

An examination of the iron-oxygen diagram¹ shows that hematite does not exist above 2651°F. It also shows that there is no liquid in the pure magnetite-hematite system until 2881°F is reached. On the other hand, in the system magnetite-wustite liquids exist at considerably lower temperatures than this. It will be seen, therefore, considering only the iron oxides, that the bonding action obtained in America in sinters comes about through considerable temperature and/or reducing conditions that produce compositions containing even less oxygen than is contained in magnetite or than results from the fusion of silicates. The bonding obtained from the iron oxides is encouraged by the reducing conditions that prevail in the vicinity of the fuel particles in a mass of this sort, where temperatures are above 2600°F.

As magnetite and wustite are opaque, they do not lend themselves to petrographic study by transmitted polarized light. The silicates found in sinter and the glass that has not crystallized transmit light and can be studied by these methods in which indices of refraction and other optical properties of anisotropic crystals lead to their definite identification. The index of refraction is the only property that can be measured in glass under the microscope, and this is a clue to its probable approximate composition.

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Discussion on this paper, TP 3673B, may be sent (2 copies) to AIME before Jan. 31, 1954. Manuscript, Feb. 24, 1953. Chicago Meeting, September 1952.

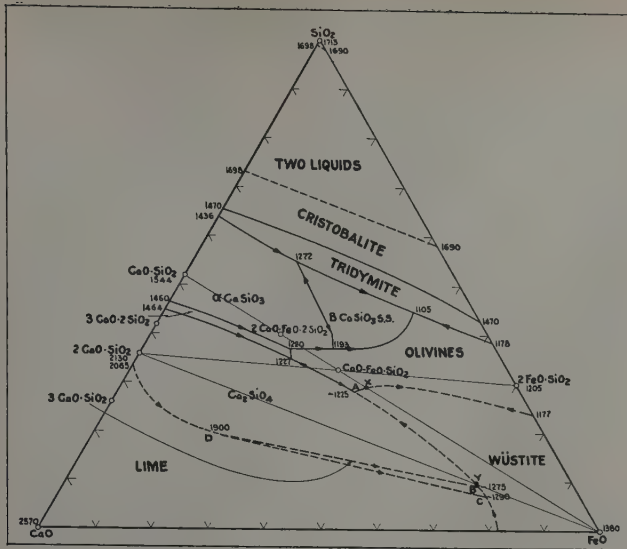


Fig. 1—Equilibrium diagram of the system CaO-FeO-SiO_2 . (After Bowen, Schairer, and Posnjak, *American Journal of Science*, September 1933, with additions by Muan and Osborn.)

Use of reflected light from the polished surface is a very helpful method of studying the structure and composition of the wustite, magnetite, and hematite. While it does not yield determinative data on the anisotropic phases or the glass, each can be identified in their broad categories as silicates and glasses and are thus distinguished from the iron oxide phase. It is possible, therefore, to study the relative proportions, distribution, and occurrence of magnetite, hematite, anisotropic silicates, and glasses in sinter structures by this means.

A petrographic examination of sinters shows the evidence of recrystallization of magnetite. Although magnetite is opaque, when it crystallizes the crystals can be seen extending into the glassy matrix. This does not necessarily mean that the temperature was sufficiently high to melt magnetite, but it does mean that magnetite had crystallized from a liquid phase, which was present during the sintering operation. Such liquids naturally include almost all the minor constituents shown in the chemical analysis of the sinter and a reasonable part of the iron oxides as well. The minor constituents, therefore, have an important bearing on the mineral constitution of the sinter. The rate of cooling and the proportion of silica, or its proportion relative to the bases present, has a marked effect upon whether crystallization takes place or solution remains as an uncrystallized bond—a glass. Although it is known that such siliceous glasses exist in the presence of basic crystals such as hercynite ($\text{FeO} \cdot \text{Al}_2\text{O}_3$), spinel ($\text{MgO} \cdot \text{Al}_2\text{O}_3$), and many others that contain no silica, the association of siliceous glasses and basic crystals might seem unusual. The fact that these glasses do not crystallize is usually evidence of a relatively high silica content, for the viscosity of siliceous liquids favor their not crystallizing. The index of refraction of these uncrystallized interstitial materials is likewise some evidence of composition to the petrographer.

These glasses are sometimes referred to as slags. This is unfortunate, as the term *slag* is so general that it is not descriptive of anything except that it is something other than the principal iron-bearing constituent of the sinter.

The presence of siliceous glass is not easy to explain in the light of our knowledge of the system

CaO-FeO-SiO_2 in which only wustite and an olivine would be expected. Although the index of refraction of an uncrystallized compound is almost always lower than that of the crystal of this composition, the glasses generally found in sinters seem to have too low an index of refraction to be uncrystallized olivine composition. It can only be assumed that they may result from local irregularities, poor mixing, inhomogeneity, or incomplete chemical reactions. It is not possible to determine the effect of Al_2O_3 in producing these glasses. The petrologist knows, however, that alumina is a glass former and encourages its formation. Heterogeneity here also is no doubt a factor in the existence of glasses in such systems.

A discussion of *slag* and glass in sinter should include the mention of fayalite. Fayalite is a mineral combining iron oxide and silica as follows: $2\text{FeO} \cdot \text{SiO}_2$. The iron oxide content is 70 pct of the compound, a much higher proportion of iron oxide tied up with silica than is ever found in a siliceous glass. In a sinter in which a high percentage of fuel is used and in which reducing conditions and high temperatures must have prevailed, it is not surprising that there is fayalite present. Much has been said regarding fayalite in connection with blast furnace operation, but despite adverse comment there is little proof of any detriment to blast furnace operation caused by fayalite in the sinter.

Fayalite melts at 2200°F . Upon contact with an adequate amount of lime at this temperature as in the blast furnace it would immediately disappear through the following reaction:



This reaction would seem to be favorable to blast furnace operation rather than unfavorable, as has been stated. According to T. L. Joseph,² fayalite is reduced in hydrogen at 1000°C (1832°F). Where this temperature exists in a blast furnace, the concentration of CO is high and no doubt between this temperature and the melting temperature of fayalite there would be good chance of its reduction taking place. If the objection of blast furnace operators to the presence of fayalite in sinter is actually directed toward excessive silica in the analysis of the sinter or excessive silica as evidenced by high percentages

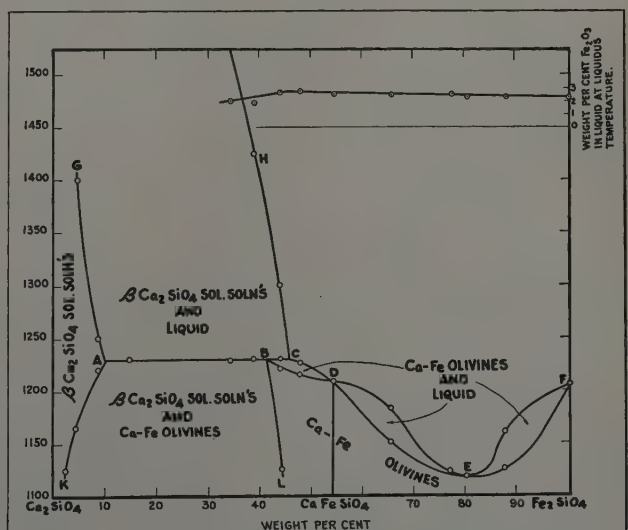


Fig. 2—Equilibrium diagram of the system $\text{Ca}_2\text{SiO}_4\text{-Fe}_2\text{SiO}_4$. (After Bowen, Schairer, and Posnjak, *American Journal of Science*, April 1933.)

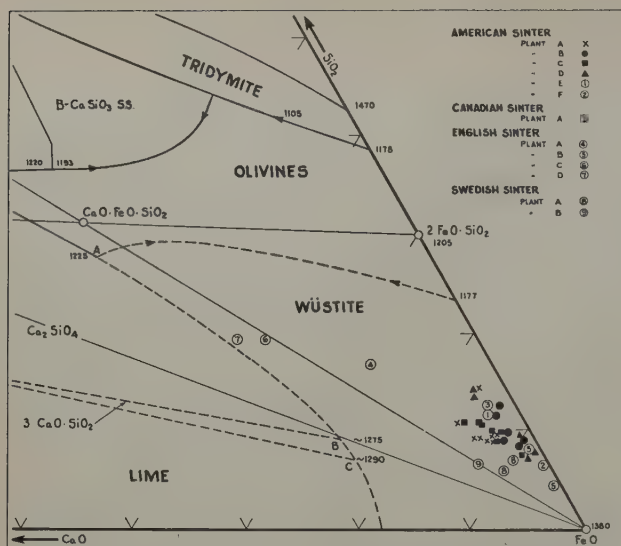


Fig. 3—Plot of sinters on CaO-FeO-SiO₂ diagram, MgO, MnO, and Fe₂O₃ included with FeO. These, with CaO and SiO₂, constituted 100 pct. Al₂O₃ omitted.

of siliceous gangue materials, the concern is acceptable and understandable.

The fayalite question should not be passed over too rapidly, and the fact that presence or absence of fayalite may be influenced by the atmospheric conditions existing in the sinter must not be neglected. When iron oxide and silica are melted under oxidizing conditions, as for example in the silica brick in the roof of an open hearth, magnetite and cristobalite or tridymite are the products found upon cooling. If fayalite is heated in air, again, magnetite and cristobalite or tridymite are formed. The presence of fayalite in sinter therefore is in itself indicative of a reducing atmosphere during melting and inadequate oxidation during cooling to form magnetite and cristobalite or tridymite. In such a case if a glass of very low index of refraction (1.46 to 1.48) were found in the sinter it could be assumed that the dissociation might have taken place without the crystallization of the appropriate silica mineral, but this has not been observed.

The question of lime is closely associated with that of fayalite. Many facets of the use of lime have been discussed. The chemistry of its use as applied to the mineralogy of sinters and slags is well known but should be discussed here.

Fayalite (2FeO·SiO₂) is the orthosilicate of ferrous iron. It is known as an olivine, a member of a group of minerals in which several other basic oxides, CaO, to a limited extent, MgO, and MnO may substitute for FeO. Lime is the most basic of these oxides, and when available in ample supply the orthosilicate of lime forms readily when proper temperatures prevail. This is the reaction which takes place when molten fayalite comes in contact with lime as described above. Where there is less lime than this available then silica combines with lime, iron, manganese, and magnesium oxide to form a more complicated olivine solid solution. As there is abundant iron oxide available in sinters, the system becomes as follows.

1—Where there is no lime, MgO, or MnO present the final product will consist of iron oxide (wustite) and fayalite.

2—With lime present but with a lime-silica ratio less than 1 the final result will be wustite and an olivine containing lime and iron oxide. (The olivine may contain also some MgO and MnO in addition to lime).

3—With a lime-silica ratio equal to 1, an olivine containing about equal proportions of lime and iron oxide (and possibly some MgO and MnO) will be present. (In this case iron oxide, manganese, and magnesium oxides are replaced to a larger extent by CaO than in point 2 above).

4—With a lime-silica ratio greater than 1 and less than 2, the minerals present in the final sinter would be wustite, an olivine approximately CaO·FeO·SiO₂, and dicalcium silicate. The "FeO" here may include some MgO and MnO.

5—With a lime-silica ratio equal to 2, the minerals present would be wustite and dicalcium silicate. The wustite would probably contain the MgO and MnO in solid solution.

According to Figs. 1 and 2, it is apparent that increasing amounts of lime beyond a certain point

Table I. Petrographic Analysis of Mineral Constituents of Sinter

Sample	Magnetite (Fe ₃ O ₄)	Hematite (Fe ₂ O ₃)	Olivine-Fayalite Silicate Series (CaO·FeO·SiO ₂ · 2 FeO·SiO ₂) Composition, Pct	Quartz (SiO ₂)	Glass, Index	Acker- manite (2 CaO·MgO· 2 SiO ₂)	Gehlenite (2 CaO·Al ₂ O ₃ · SiO ₂)	Siderite (FeCO ₃)	Calcite (CaCO ₃)	Total Carbon in Raw Mix, Pct
A*	M	m	m (FeS) 65 (C ₂ S) 35	m	m 1.645	o		~	~	3.95 to 4.22
B*	M	s	L (FeS) 95 (C ₂ S) 5	m	s 1.648	~	~	~	~	6.22 to 6.62
C*	M	s	L (FeS) 85 (C ₂ S) 15	s	s 1.643	~	~	~	~	4.65
D†	M	m	m (FeS) 80 (C ₂ S) 20	L	m 1.658	~	~	~	~	
E**	M	m	Possible s	L	m 1.675	~	m	m	~	
F**	M	s	Possible s	m	m 1.645	~	~	~	~	
G‡	M	L	m (FeS) 60 (C ₂ S) 40	m	m 1.620	~	~	~	~	
H‡	M	L	m (FeS) 55 (C ₂ S) 45	m	m 1.654	~	~	~	s	

* Bethlehem Steel Co. sinter.

† Another American sinter.

** English sinter.

‡ Swedish sinter.

FeS = 2 FeO·SiO₂ (fayalite).

C₂S = 2 CaO·SiO₂ (dicalcium silicate).

M = Major constituent.

L = Large amount.

m = Moderate amount.

s = Small amount.

o = Occasional grains.

would have the general effect of raising the melting temperature of the olivine phase which includes the minor constituents. In fact, when the lime-silica ratio is equal to 2, the wustite-dicalcium silicate combination is very refractory and *dry* and may be quite weak. It might also *dust* because of the inversion of dicalcium silicate from its beta to gamma form. Dusting is therefore not to be mistaken for slaking of lime. Slaking of lime has been known to occur in sinter mixtures wherein the lime or limestone addition was too coarse-grained to become fluxed, and only calcined during the sintering operation.

The System, CaO-FeO-SiO₂, by Bowen, Schairer, and Posnjak,³ illustrates the trend toward higher melting temperature in this system in the direction of dicalcium silicate and lime.*

* The authors have developed a lucid method of presentation showing the areas which are liquid at various temperatures. The diagrams are of particular interest to anyone interested in the chemistry of sinter.

Fig. 3 shows a plot of a number of sinters on the CaO-FeO-SiO₂ diagram. These sinter compositions were calculated to a 100 pct basis with the MgO and MnO included in the FeO. Only CaO, SiO₂, FeO, MnO, and MgO are included. If a line is drawn from the FeO corner through any of these sinter compositions, the point where the line intersects the CaO-FeO-SiO₂-fayalite join represents the composition of the olivine that should exist in the sinter. The lime-silica ratio of the sinter composition controls the point of intersection and therefore the composition of the olivine phase. The sinters plotted on Fig. 3 seem to vary in composition only slightly, but it will be apparent when one follows the above use of the phase diagram that sinters containing different kinds of olivine as the silicate phase will be found. Table I shows what was actually found in sinters examined at Bethlehem Steel Co. Table II is a chemical analysis of sinters reported in Table I.

Table II. Chemical Analysis of Sinters Reported in Table I

Chemical Constituents, Pct	A*	B*	C*	D†	E**	F**	G‡	H‡
Total Fe	56.16	59.60	58.04	59.00	35.80	64.00	61.10	57.70
SiO ₂	8.84	7.92	10.84	17.70	16.30	7.15	5.35	6.00
CaO	3.68	0.20	1.26	3.60	18.40	0.66	4.10	5.80
MgO	2.08	0.48	0.39	1.30	2.16	0.50	1.35	3.13
MnO	0.86	0.39	0.99		1.20	0.30	1.55	1.36
S	0.01	0.01	0.02		0.15	0.01	0.02	0.01

* Bethlehem Steel Co. sinters.

† Another American sinter.

** English sinter.

‡ Swedish sinter.

Discussion of the magnetic properties of sinters must not be omitted. Not all magnetic iron is magnetite. In sinters produced from magnetite concentrates it is possible to have magnetic gamma Fe₂O₃ maghemite. Care should always be exercised in calling such magnetic constituents magnetite solely on the basis of a magnetic test.

In conclusion, it should be emphasized that while details of the foregoing discussion of the mineralogy of sinter have been verified by petrographic examination in sinters of which samples were available, sinter is a more or less heterogeneous result of relatively incomplete chemical reactions. This must always be borne in mind when the mineralogy of a sinter is being discussed, since there may be other factors having equally important bearing on physical properties.

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Discussion*

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F — Coal

- Cleaning Various Coals in a Drum-Type Dense-Medium Pilot Plant. (Paper by M. R. Geer, W. A. Olds, and H. F. Yancey, *Transactions AIME*, **196**, 696; *Mining Engineering*. July 1953. Discussions by J. S. Huckaba and N. L. Davis.)..... 1125
- The Blending of Western Coals for the Production of Metallurgical Coke. (Paper by John D. Price, *Transactions AIME*, **196**, 716; *Mining Engineering*. July 1953. Discussion by R. W. Campbell.).. 1127
- Two-Way Belt Conveyor Transportation. (Paper by C. W. Thompson, *Transactions AIME*, **196**, 905; *Mining Engineering*. September 1953. Discussion by Paul D. Suloff.)..... 1128

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I — Mining Geology

- The Role of Geologists in the Development of the Labrador-Quebec Iron Ore Districts. (Paper by J. K. Gustafson and A. E. Moss, *Transactions AIME*, **196**, 593; *Mining Engineering*. June 1953. Discussion by A. K. Snelgrove.)..... 1129

L — Geophysics

- Geophysical Case History, Fredericktown Lead District. (Paper by Harold Powers, LeRoy Scharon, and Carl Tolman, *Transactions AIME*, **196**, 317; *Mining Engineering*. March 1953. Discussion by Sherwin F. Kelly.)..... 1130

* TP 3674

A — Metal Mining

An Analysis of Mine Opening Failure by Means of Models

by Bernard York and John J. Reed

DISCUSSION

Louis A. Panek (*College Park, Md.*)—The authors have chosen a problem that merits considerable study, because the phenomenon of failure of rock in the vicinity of an underground opening is important, although far from being adequately understood.

It is interesting to find that the model cracks appear to coincide with the trajectories of maximum shearing stress when the opening exists in a body subjected to hydrostatic pressure. It would be instructive to determine whether this is true for unidirectional loading as well.

It may be true, under certain conditions, that the larger the opening in rock or a similar material, the weaker the opening. The writer cannot agree, however, that this conclusion is supported by the experimental results. Fig. 12 suggests that this rule does not apply to openings more than about 1.5 in. in diam. Since a 1.5-in. opening has little importance in mining, if Fig. 12 is to be capable of practical application, it must be considered to give the results of testing scale models that represent the behavior of much larger openings, and the well known requirements as to similarity between a model and its prototype must be satisfied. To calculate the size of mine opening represented by such a model it is necessary, in general, to determine the scale ratio (ratio of a prototype dimension to the corresponding model dimension). The conclusion that "the larger the opening the weaker it is" implies that this is true for openings of any size, i.e. that this holds true for all values of the scale ratio. By arbitrarily choosing scale ratios of 240 and 384 and model open-

ings of $\frac{3}{4}$ in. and 1 in., as shown in the accompanying table, it can be "proved" that since a 1-in. model open-

Size of Model Opening	Size of Prototype Opening	
	Scale ratio = 240	Scale ratio = 384
$\frac{3}{4}$ in. 1	15 ft 20	24 ft 32

ing is weaker than a $\frac{3}{4}$ -in. opening, therefore a 20-ft mine opening is weaker than a 24-ft mine opening. This obvious inconsistency serves to illustrate the fact that actually the scale ratio may not be arbitrarily chosen. If the scale ratio is indeterminate, as is apparently the case, it is impossible to specify the size of mine opening represented by any model hole smaller than 1.5 in.; hence the results obtained for model holes smaller than 1.5 in. cannot be applied to mine openings. Fig. 12, therefore, indicates that the model holes should be at least 2 in. in diam in order to represent mine openings; a free choice of scale ratio can then be made, since size will have no effect on the strength of the opening. Moreover, with the larger opening a difference in strength between a square and a circular shape may be exhibited. An alternative is to determine the dimensional relationship between "the incipient cracks or other weaknesses in the (model) material" and those in the mine structure, and determine the scale ratio from this relationship. The size of mine opening represented by any model hole could then be calculated.

B — Minerals Beneficiation

Magnetic Roasting of Lean Ores

by Fred D. DeVaney

DISCUSSION

W. O. Philbrook (*Carnegie Institute of Technology, Phila.*)—Mr. DeVaney's paper is a valuable addition to the growing technology of the magnetic roasting of lean iron ores. His furnace is ingeniously designed to permit close control of the process and to achieve high thermal efficiency by counter-current heat exchange and the withdrawal of both solid and gaseous products at relatively low temperatures. The principle of using a lean reducing gas to avoid over-reduction of the ore even at high operating temperatures (for rapid reaction rate) is thoroughly sound. The gas analyses and flow rates reported, however, are internally incon-

sistent and render suspect any details of a heat balance based upon them. While it is entirely proper to consider the entrant gas as a reagent and to account only for the heat evolved by consuming a portion of the gas, many operators are accustomed to a heat balance using the total calorific value of the reagent supplied; and comparison of heat balances made with different bases of calculation can be quite misleading. Since Mr. DeVaney's paper is likely to be cited frequently in future discussions of magnetic roasting processes, it seems worth while to point out the internal discrepancies and to give alternate values of heat requirements for different bases of calculation.

The statement is made on p. 1222 that "because of air leakage 32.6 pct of the gas was burned in the furnace," and a heat evolution of 126,600 Btu per ton of feed was attributed to this combustion. This assumption of air leakage was required to obtain a CO balance for the reported analyses and flow rates of entrant and exit gases, but it will be shown that it cannot be supported by total material balances for individual chemical elements. Furthermore, it seems physically impossible that there should have been any significant air leakage into a furnace operating above atmospheric pressure, and the reported air supply to burn the fuel oil is, if anything, a trifle less than the theoretical requirement. The premise that reagent gas was burned by air within the furnace cannot be proved by the evidence available.

Before the inconsistencies of the data may be demonstrated, it is necessary to make some reasonable assumptions on points not clearly defined in the paper. "Standard conditions" for gas volumes are assumed to be 60°F, 30 in. mercury pressure, dry, under which conditions a pound mol of perfect gas occupies 379 cu ft. On this basis, the amount of CO required to reduce a pound of iron in the form of hematite to magnetite is 1.13 cu ft, instead of 112 cu ft as stated on p. 1222. The analyses and flow rates of the exit gases shown in Fig. 4 of the paper are ambiguous in that a wet analysis is given for gases leaving the furnace, but the flow rate for the moist recirculated gas is the same as given for dry recirculated gas leaving the scrubber and entering the bottom of the furnace, namely 385 cfm. It will be assumed that reported flow rates refer to dry gases in all instances.

To be consistent with the thermal data reported, the fuel oil is assumed to have the following properties: 30° API, 7.29 lb per gal; heating value 19,450 Btu per lb (gross); 18,310 Btu per lb (net), 133,500 Btu per gal (net). A reasonable analysis would be 86 pct C, 12 pct H, 2 pct O + N + S + ash. On this basis the theoretical air requirement for 1.5 gal per hr of oil is 5.30 mols per hr or 33.5 cfm (as compared with 33 cfm reported), and the theoretical products of combustion are 0.78 mols CO₂, 0.66 mols H₂O, and 4.19 mols N₂.

The iron content of the feed and products, calculated from the weight and analysis of the roasted ore, is taken to be 650 lb Fe per hr. This is equivalent to 929 lb of Fe₂O₃ in the feed, 898 lb of Fe₂O₄ in the product, and 31 lb O or the equivalent of 0.97 mols O₂ removed by the roasting process. The moisture evaporated from the feed is 78.4 lb or 4.35 mols H₂O per hr.

Material balances may now be made for total quantity of gases and for individual elements. The recirculated gas may be dropped from the balances, except for moisture calculation, since Fig. 4 shows the same volumes and same dry analyses of recirculated gas leaving the top and entering the bottom of the furnace. It should be noted that there is no change in volume when CO is oxidized to CO₂ by oxygen from Fe₂O₃, but oxidation of CO by air would entail the introduction into the gas of N₂ equivalent to 3.76 times the volume of O₂ consumed or 1.88 times the volume of CO converted to CO₂. For a perfect gas (at 60°F, 30 in. Hg, dry) mols per hr = cfm × 60/379, and the individual constituents are calculated from gas analyses and flow rates. Equivalent mols C = mols CO + mols CO₂, and equivalent mols O₂ in gas = ½ mols CO + mols CO₂. An element balance is given in Table II in which the reported gas analyses and flow rates are used as the basis of calculation, ignoring any leakage air and assuming theoretical products of combustion of oil.

Table II shows that the vented stack gas is too low in C, O₂, and H₂O to account for the input, but far too high in N₂. The addition of leakage air to the input to bring the N₂ figures into balance would throw the oxygen balance still farther out of line. Without giving the detailed calculations, it is sufficient to say that a balance for N₂ requires 7.18 mols of leakage air; a balance for the volumetric flow of entering gases to be consistent with the volumetric flow rate of stack gas (both on a dry basis) requires 5.7 mols of leakage air;

and a CO balance (ignoring discrepancies in the C balance) requires 2.74 mols of leakage air. If the gas analyses were internally consistent, it would be possible to solve simultaneous equations for C, O₂ and N₂ balances to obtain the volumes of producer gas, air, and stack gas, but such an attempt leads to inconsistent and ridiculous answers. Obviously the gas analyses and reported flow rates are seriously inconsistent and will not support any material or heat balance calculations based upon them.

Table II. Element Balances, Mols per Hr

Input	CO	CO ₂	Equiv. C	Equiv. O ₂ *	H ₂ O	N ₂
Producer gas	3.39	0.31	3.70	2.01		6.59
Products of comb. of oil		0.78	0.78	0.78	0.66	4.19
Ore, drying + reduction				0.97	4.35	
Total			4.48	3.76	5.01	10.78
Output						
Stack gas	0.30	3.05	3.35	3.20	4.25†	16.45

* Equivalent O₂ in carbon oxides and from ore, but not including H₂O.

† Assuming 510 cfm dry exit gas, 5 pct H₂O on moist basis:

$$\frac{510}{0.95} \times 0.05 \times \frac{60}{379} = 4.25 \text{ mols H}_2\text{O}.$$

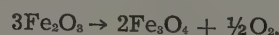
Turning now to heat balance calculations, it is evident that an item for heat released by burning a portion of the entrant gas within the furnace is, at best, uncertain. There is little justification for a detailed heat balance including sensible heats in entrant and exit gases because the volumes of the gases involved are not reliable. This is not a serious defect, however, because the heat contents of entering gases seem to be a by-product of the energy of compressing them, and the temperature of the exit materials is so low that further recovery of heat from them would be impracticable. It should be sufficient for practical purposes to characterize the heat balance by the external heat required.

No heat balance can be properly calculated until the system and the basis for calculation have been defined. It is quite proper to limit the system to the furnace itself and to consider the producer gas as a chemical reagent (not as a fuel) as Mr. DeVaney has done. In this event, however, the exothermic heat of the reaction



should properly appear in the "input" side of the heat balance to account for reagent consumed. Using the best available data,¹ the ΔH of this reaction is -12.1, kcal or 21,880 Btu evolved for 2 lb mols Fe₂O₃, which amounts to 47.2 Btu per lb Fe₂O₃ formed, instead of 88 Btu as stated by DeVaney on p. 1220. A balance which includes only the net heat actually evolved in the process is the best for analyzing the distribution of heat within the process, but it may not be suitable for comparison of one process with another for which a different basis of calculation was used.

In order to compare the heat requirements of the DeVaney process with some other type of reducing roast for which the total heating value of the reagent has been considered as "heat supplied," it is necessary to put the thermal calculations all on the same basis. One possibility is to define the system to comprise the furnace and oil burner and to include the calorific value of the producer gas as well as of the oil in heat supplied. Another possibility is to include the gas producer within the system and consequently to consider the calorific value of the coal or coke fed to the producer as heat supplied. In either event, since the calculation implicitly assumes burning all C or CO to CO₂ in some fashion, the appropriate heat of reaction is for



which is endothermic to the extent of 216 Btu per lb of Fe₂O₃ and should appear under "heat consumed."

The heat input to the DeVaney process according to each of the three methods outlined is given below. It has been necessary to accept the analysis of the producer gas and the flow rate of 65 cfm given in Fig. 4,

Table III. Heat Supplied to DeVaney Process

For 1 ton of feed; sensible heats neglected; base temperature 25 C (77°F); H ₂ O (g) as standard state.				
Basis A: System comprises furnace only. Producer gas is treated as reagent without regard to its calorific value.				
Heating value (net) of oil: 1.5 x 133,500				200,000 Btu
Heat of reaction: 47 Btu per lb Fe ₃ O ₄ x 898 lb Fe ₃ O ₄				42,000
Total input				242,000 Btu
Basis B: System comprises furnace only; calorific value of producer gas as a fuel considered as a cost of the process.				
Heating value (net) of oil: 1.5 x 133,500				200,000 Btu
Heating value of producer gas:				
65 ft ³ per min x 60 min per hr x 0.33				
CO x 321 Btu per ft ³				410,000
Total input				610,000 Btu
Basis C: System includes gas producer; calorific value of coke fed to producer considered as a cost of the process.				
Heating value (net) of oil: 1.5 x 133,500				200,000 Btu
Heating value of carbon to producer:				
65x60 mols gas	x	0.36 mols C	12 lb C	14,600 Btu
379	hr	mol gas	mol C	lb C in coke
Total input				850,000 Btu

and the amount of carbon fed to the gas producer has been calculated because of the absence of any statement of coal or coke consumption. The calculated thermal requirements are rather uncertain, therefore, but they are the best that can be obtained from the experimental data supplied.

The difference between Bases A and B in Table III rests in a different viewpoint as to "heat of reaction" plus heating value of unconsumed CO in the stack gas. The difference between Bases B and C represents partial combustion of C to CO and CO₂ in the gas producer plus heat losses in the producer. The choice of basis depends upon the use to be made of the results.

It is not the intention of this discussion to discredit Mr. DeVaney's contribution. Considering the difficulties of metering gas flows accurately under conditions of varying temperature, pressure, and composition, and the low precision often inherent in Orsat analyses, it is a difficult task to obtain valid material balances under the conditions of this process. It is hoped that this discussion may add to the inherent worth of Mr. DeVaney's paper by giving a clearer perspective for comparing it with other magnetic roasting processes.

¹ U. S. Bureau of Standards: *Circular 500, Selected Values of Chemical Thermodynamic Properties* (Feb. 1, 1952).

Effects of Alkalinity on the Flotation of Lead Minerals

by Marston G. Fleming

DISCUSSION

E. C. Peterson (*Anaconda Copper Mining Co., Darwin, Calif.*)—A study of this quite comprehensible and interesting paper by Dr. Fleming brings to mind several observations in the practical application of alkalinity and related factors to the actual practice of lead mineral flotation.

Soda ash has long been a widely used and a very helpful alkaline conditioning agent in the flotation of galena from the usual run of lead-zinc ores. Soda ash is one of the common "standard" conditioning agents tried in any laboratory investigation of lead-zinc ores because it has so often proved helpful to galena flotation. However, the use of soda ash in the actual flotation of oxidized lead ores is certainly not widespread.

In the flotation of certain lead-zinc ores from the Park City district of Utah, it was found in the usual cyanide-zinc sulphate-xanthate circuit that soda ash had an effect of producing a very condensed and flat froth regardless of the many frothers tried and that substitution of lime for soda ash to produce the same pH (8.0 to 8.4) improved the froth condition. However, the flotation of coarse particles of galena became so critical that some would pass through into the tailing, but caustic soda as a substitute for either soda ash or lime produced a very desirable froth condition in the same pH range and greatly improved the metallurgical outcome. Milling was carried out in typically "hard water" from watersheds of limestone and other calcareous rocks and the ore also apparently contributed magnesium and calcium salts to the pulp.

Certain lead-zinc ores of Mexico have shown their greatest flotation response in circuits conditioned with sodium bicarbonate, and such actual mill use was also believed to be related to the water necessarily used in milling. On other lead-zinc ores the optimum has been obtained in actual milling treatment by use of caustic soda in both circuits of the operation.

In flotation of oxidized lead ores in which sulphidization is employed by addition of sodium sulphide, very high pH's exist in the flotation circuit. With the usual oxide lead ores, demanding 5 to 15 lb of sodium

sulphide per ton of ore, pH's in the circuit (closed and open water-circuits) may range from 9.5 to 12, and for ores that contain much anglesite or a high percentage of iron oxide minerals, sodium sulphide consumption may reach as much as 30 lb per ton of ore and the resultant pH will be correspondingly higher. Yet in such flotation treatment employing either xanthate or oil (high-sulphur crude oil or diesel oils) as the collector, satisfactory to excellent grades of concentrate and lead recovery are obtained.

Although there have been great studies and accomplishments in the investigations of flotation fundamentals and flotation theory, and although an investigation to determine the effect of all factors entering into the actual plant flotation of sulphide and oxide ores must become very complicated, as Mr. Fleming has mentioned in his caution regarding the drawing of general conclusions for other ores, it seems that it would be of greater interest if the techniques and investigations could, with time, approach the conditions of actual practice and lead the way to improved and more efficient flotation plant performance.

Marston G. Fleming (author's reply)—The gulf between fundamental flotation research and plant operation is, perhaps, less wide than Mr. Peterson suggests. For example, the work described in my paper developed from an extensive investigation of a complex lead-vanadium ore from southwest Africa. This investigation started as a standard ore-testing problem but, at almost every stage, results were obtained which could not be interpreted in terms of previous experience. A program of fundamental research was therefore undertaken and was closely interlocked with the ore testing. This coordinated investigation resulted in a flotation process which has now been proved by two years of successful plant operation, and although the grade and mineralogical constitution of ore as well as smelter requirements have altered more than could have been anticipated, our knowledge of the fundamental character of the problem has made it possible to meet each change in conditions with much more confidence than would have been the case had we neglected this aspect of the investigation.

Upgrading of Low Grade Manganese Ores by Leaching with Caustic Soda

by R. V. Lundquist

DISCUSSION

L. A. Roe (*Bjorksten Research Laboratories, Inc., Madison, Wis.*)—This paper is typical of the current trend of thought in the mineral beneficiation industry. As we focus our attention on the more complex and the lower grade ores we find, in many cases, that older methods of beneficiation are either too costly or do not apply at all. With the exception of methods of comminution, there is little hope that we can greatly increase efficiencies of the various unit processes currently used to beneficiate ores. Our great hope is in new methods of beneficiation, and heading this list are "chemical beneficiation 'methods.'" It is often said that the inorganic chemists have failed to keep pace with the organic chemists. When we observe the progress of the plastics, petroleum, and medicinal fields there seems to be considerable evidence that this is true. Now when we turn to the mineral beneficiation industry it is only too evident that we have failed to keep pace with either the organic or the inorganic chemists in applying new chemical theories and techniques to our field. Our main excuse has been that we cannot afford these expensive chemical methods except in the case of the rare and precious metal ores. With the advent of large-scale processing of low-grade magnesium, aluminum, nickel, and uranium ores by chemical

methods it is to be hoped that in the near future we will see the spread of similar methods to process iron, tin, manganese, and many other ores.

I wish to offer information which will supplement the evidence presented by this paper as regards the value of caustic leaching of silica from ores. Using low-grade Michigan iron formation composed chiefly of goethite and chert, I have applied the principles set forth in this paper with the exception that higher pressures and temperatures were used. The results listed below show how effectively this chemical beneficiation method can upgrade iron ores:

	Fe, Pct	SiO ₂ , Pct
Original iron formation	36.09	44.72
Processed iron concentrate	60.33	6.54

The crude ore used in the above test was ground in a ball mill until the ore was 85 pct —200 mesh. The leaching operation was carried out at 200°C and 200 lb per sq in. of pressure in a cast-iron autoclave. Only one stage of leaching was used. The potentialities of the method in producing extremely high grade iron concentrates for use in the production of powdered iron are great.

Measurement and Evaluation of the Rate of Flotation as a Function of Particle Size

by T. M. Morris

DISCUSSION

R. T. Hukki (*Finland Institute of Technology, Helsinki, Finland*)—Two schools of thought are developing concerning the order of the rate of flotation. The paper by Morris is intended to show experimental evidence in favor of the first order rate.

Arbiter, on the other hand, has presented an earlier paper¹⁰ to demonstrate the validity of the second order rate equation. Unfortunately his method of plotting is open to criticism, as pointed out by Morris.¹¹ A more sensitive method of plotting has been described by the writer,¹² whereby the data presented by Arbiter demonstrate the superiority of the second order plot.

The general rate equation may be written in the following forms for the integrated first and second order equations:

$$(A) \quad t = -\frac{1}{k} \ln \frac{C_0 - R}{C_0} \quad (\text{first order}) \quad [4]$$

$$(B) \quad \frac{1}{t} = -kC_0 + kC_0^2 \frac{1}{R} \quad (\text{second order}) \quad [5]$$

As a first approximation the value of C_0 may be taken as a unity, 100 pct or 1. Whether or not this is the case in all tests performed by Morris is impossible to judge, owing to the fact that the flotation times are extremely short, e.g. 150 sec in tests A. It would, indeed, be very interesting to know what the total recovery R would be after 1 hr of flotation, for example, instead of after 150 sec. Certainly there are circumstances when C_0 approaches unity within a very small

margin of error. Morris has used the same approximation in his example D "since the feed to this cleaner

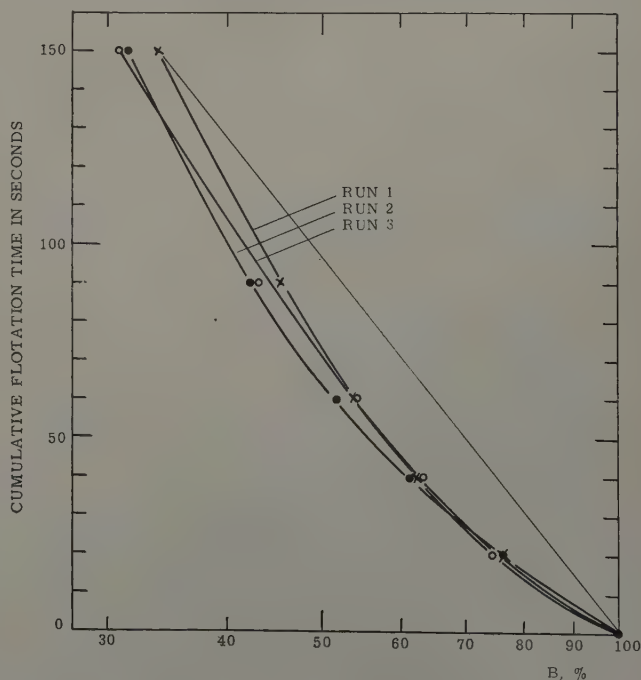


Fig. 8—First order plots according to Eq. 6 of the uncorrected data by Morris.

machine consisted of particles which had floated in the preceding rougher operation." In tests A he used "cleaned concentrate made from Tennessee Copper Co. ore." No information was given as to the prehistory of the method of cleaning. In this particular case, however, Morris has had to use a factor x to prove what he intends to prove.

Morris bases his thesis on the idea that flotation rate data must give a straight line by the method of plotting he adopts. To make the method work, he needs the crutches provided by factor x . Factor x , the value of which is omitted except in one example, varies according to numerous variables. It must be found by trial and error. By use of proper scales, the plots finally appear relatively straight.

The scientific validity of the method is something to question. By exercise of similar patience the second order plot given by Eq. 5 can also be shown to give a straight line with selected data and with proper factor x . In the writer's opinion this proves nothing.

An interesting feature in the paper by Morris is the study of flotation rates in relation to size fractions. At the same time, however, he overlooks the difference in relative floatabilities of pyrite and pyrrhotite and reports the final recovery in grams in tests A. Perhaps the method of analysis described by McLachlan¹³ would have been an additional useful tool in the evaluation of products in tests A.

To demonstrate the limitations of the method presented by Morris the writer would like to present a re-examination of part of the evidence cited by Morris.

First order plot: The first approximation of Eq. 4 may be written as

$$t = -\frac{1}{k} \ln(1-R) = -\frac{1}{k} \ln B \quad [6]$$

It is seen from Eq. 6 that graphical representation of flotation time vs recovery in tailing ($1-R$ or B) should form on a semilogarithmic paper a straight line passing through the point $t = 0, B = 1$ providing the flotation process follows the first degree rate equation.

Accompanying Table V gives the original data by Morris (his Table I) and Fig. 8 shows the respective first order plots.

Second order plot: By giving factor A again the value of unity Eq. 2 can be given as

$$\frac{1}{t} = -k + k \frac{1}{R} \quad [7]$$

or

$$1 + kt = \frac{1}{B} \quad [8]$$

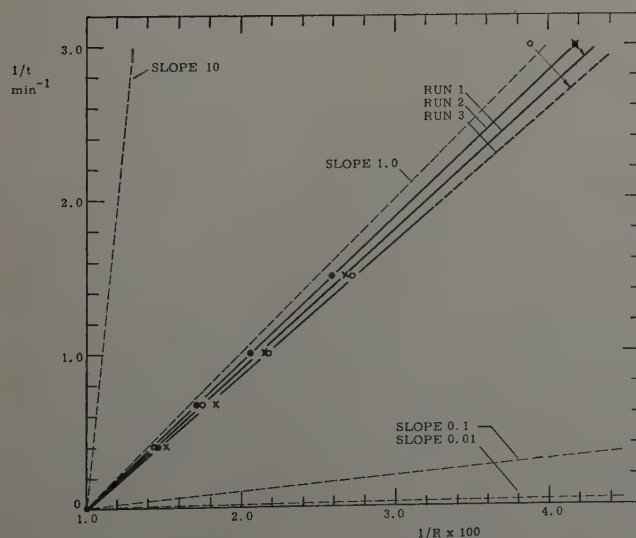


Fig. 9—Second order plots according to Eq. 7 of the uncorrected data by Morris.

It is seen from Eq. 7 that as the flotation time becomes infinite, the ratio $1/t$ becomes 0 and the ratio $1/R$ becomes 1. Thus the experimental flotation results, when plotted as $1/t$ against $1/R$, should form a straight line passing through the point $1/t = 0, 1/R = 1$, intercepting the 0—ordinate at the negative value of the rate constant and having the slope of the same positive numerical value as the intercept, all this under the condition that the flotation process follows the second degree rate equation. Eq. 8 indicates that by plotting the reciprocal of the diminishing recovery of the valuable mineral in tailing against cumulative flotation time, a straight line relationship should again be obtained. This method of plotting is more sensitive than the preceding one toward the end of flotation.

Data by Morris for graphical presentation according to Eq. 7 are given in Table VI and according to Eq. 8 in Table V. The respective plots are shown in Figs. 9 and 10.

The evidence given above indicates:

1—The data do not satisfy the requirements of the first order rate equation, because the plots are definite curves.

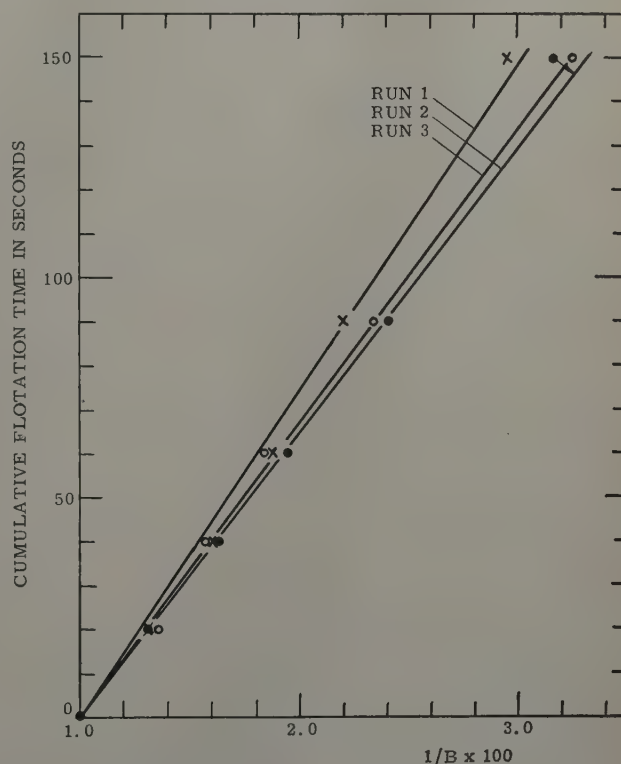


Fig. 10—Second order plots according to Eq. 8 of the uncorrected data by Morris.

2—In Fig. 9 Run 2 gives an excellent straight line according to Eq. 7. The first points of Run 1 and Run 3 are off the straight line. It is reasonable to point out the possibility of experimental errors owing to the fact that the three tests are parallel.

3—Run 3, on the other hand, gives a good straight line in Fig. 10. Only small deviations chargeable to experimental errors can be found in the other two.

Table V. Results of Test A

Pan	Cumulative Time, Sec	Cumulative Weight of Tailing, Pct (B)			1/B x 100		
		Run 1	Run 2	Run 3	Run 1	Run 2	Run 3
1	20	76.1	76.1	74.2	1.31	1.31	1.35
2	40	62.6	61.3	63.2	1.60	1.63	1.58
3	60	53.5	51.4	54.0	1.87	1.94	1.85
4	90	45.4	41.6	42.7	2.20	2.40	2.34
5	150	33.9	31.6	30.8	2.95	3.16	3.25

Table VI. Results of Test A

Pan	Cumulative Time, 1/t, min ⁻¹		Cumulative Weight, Pct (R)			1/R x 100		
	Sec	min ⁻¹	Run 1	Run 2	Run 3	Run 1	Run 2	Run 3
1	20	3	23.9	23.9	25.8	4.18	4.18	3.88
2	40	1.5	37.4	38.7	36.8	2.67	2.59	2.72
3	60	1	46.5	48.6	46.0	2.15	2.06	2.17
4	90	0.67	54.6	58.4	57.3	1.83	1.71	1.74
5	150	0.40	66.1	68.4	69.2	1.51	1.46	1.44

4—Data in Table II by Morris give evidence similar to that discussed above. The finer fractions show less perfect plots, because of cumulative errors. This is not visible in figures prepared by Morris.

5—Longer flotation times are highly desirable to demonstrate the true total recovery.

From previous experience the writer would like to point out that:

1—He has not found a single set of data to give reliable support for the first order rate equation.

2—He has found about one third of all experimental evidence available in published results by other investigators favorable for the second order relationship.

3—The slope of the second order plot, Fig. 9, is a useful *flotation rate index*, the numerical value of which increases with improving performance, e.g., 0.01 for very slow; 0.1 for slow; 1 for normal; and 10 for very fast flotation.

4—Flotation rate tests should be conducted with automatic flotation apparatus to be dependable. Only then can the statement be justified that "the real test of the graphical method is whether or not the earliest points agree with a straight line."¹¹

5—Flotation rate studies may never be conclusive enough to prove exclusively the validity of the first or second order rate process.

¹⁰ N. Arbiter: *Trans. AIME* (1951) 190, p. 791.

¹¹ T. M. Morris: *Trans. AIME* (1951) 190, p. 991.

¹² R. T. Hukki: *Flotation Rate and Flotation Rate Index. Suomen Kemistilehti* (1952) B 25, pp. 29-35.

¹³ C. G. McLachlan: *Trans. AIME* (1934) 112, pp. 570-596.

T. M. Morris (author's reply)—The remarks made by Mr. Hukki are appreciated. His main criticism seems to involve my use of a factor to evaluate the percentage of floatable material in a flotation operation.

The law of mass action states that the "rate of a chemical reaction is proportional to the 'active mass' of the reacting substances present at any given time."

The determination of active mass is necessary not only in rate studies, but also in thermodynamic calculations where the term activity is used in place of active mass. Therefore it is no more permissible to assume all of the mass as active in a rate study than it would be to assume activity to be unity in a thermodynamic calculation.

Hence any analysis of flotation rates which assumes that all of the solid particles will float is unrealistic and the law of mass action is violated.

Probably the most practical method for evaluating the percentage of floatable solids is to continue the flotation operation until no particles of the mineral to be floated report in the froth. In a batch cell this method must sometimes be used with caution as explained on page 797 of my paper.

I agree with Mr. Hukki that graphical presentation of flotation rate studies probably will not be conclusive enough to prove the order of the rate. The use of a flotation rate equation should prove a useful tool in the study of flotation operations.

Sampling and Testing of Sinter

by R. L. Stephenson and D. J. Carney

DISCUSSION

M. O. Holowaty (*Inland Steel Co., Chicago*)—Messrs. Stephenson and Carney should be commended on the presentation of this very interesting paper. They have introduced new ideas on sampling of sinter and have thereby made a valuable contribution to the qualitative evaluation of sinter. The paper undoubtedly presents a novel approach to the problem of sinter sampling and this is to be regarded as its main value.

Sampling of sinter from the pallets prior to the discharge is by no means new. This sampling procedure was used in the past by Philbrook,³ Hamilton and Ameen⁴ and others. The sampling basket used by the authors of this paper, however, permits one actually to obtain a good sample of the sintering mixture. The basket which can also be described as a removable pallet goes then through the entire process of sintering and thus offers a sample which is positively representative of the batch in question.

The authors' method of evaluating physical properties of sinter seems to be not altogether clear, however. It is felt that the following points should be clarified:

1—In the section describing the shatter test the authors state that a 5-lb sample was used in the actual tests. There is no mention of the initial size of sinter particles used and the method of obtaining this 5-lb sample from the sinter cake weighing about 100 to 120 lb. It is also surprising that the sample was dropped from a height of 3 ft instead of the conventionally used 6 ft. Unfortunately there is no mention

of the number of drops to which the sample was subjected.

2—The tumbler test used by Messrs. Stephenson and Carney in testing the samples of iron ore sinter was performed in a standard ASTM coke tumbling drum. The drum measures 3 ft in diam and is equipped with two 2-in. lifter bars. The distance of fall in this testing drum is identical with the distance used in the previously described shatter test. The number of drops in the tumbling drum is approximately double the number of revolutions. The basic difference then between the drop-shatter test used by Stephenson and Carney and the tumbling test seems to be the number of drops to which the sample is subjected and the abrasive action to which the sinter particles are exposed in the rotating drum. In any case, the choice between the shatter test and the tumbler test seems to be only a matter of taste and convenience.

3—The authors of the paper state that "in order to insure getting results that were representative . . . the entire sample obtained by means of the sampling basket was used as the starting material for the tumbler test. If the sample broke in pieces, all the pieces were used regardless of their size."

The initial size of the material is known to be a very important factor, influencing seriously the final size of the particles. This should be particularly important in case of hard sinters. It seems also very troublesome and inconvenient to charge 12-in. cubes of sinter weighing about 100 lb into the testing drum. This could be eliminated by taking a standard fraction of somewhat smaller particles.

4—Messrs. Stephenson and Carney use Power's method to express the results of sieve analyses used in the evaluation of the material degradation following the strength test. Power's method involves plotting the size of the sieve openings on a logarithmic scale and the cumulative percentage of material retained on each screen on the probability scale. The median size is derived from the graph by interpolating the plotted data and calculating the opening size of the screen on which 50 pct of particles would be retained.⁵

This method has serious disadvantages. Let us consider, for example, two sinters, A and B. A is a medium hard sinter. In the strength test it breaks up into particles of various sizes with values evenly distributed over the whole range on the graph of the screen analysis. B is a spotty, partially burnt sinter. The product of the strength test performed on this sinter contains a small percentage of hard, slaggy lumps and a high percentage of unagglomerated fine material. The plotted values of the screen analysis shows in this case no change in cumulative percentage of material retained on the succeeding screens. The median sizes derived from the plots of the screen analyses of the two products may then give identical values without revealing the difference in the physical structure of the material.

It seems obvious that these two sinters cannot possess the same strength and that their performance in the blast furnace will be entirely different. It would seem advisable to express the results of a strength test in another clearer way.

The method used by Voice, Lang, and Gledhill, for instance, seems to be quite convenient.⁶ According to this method the strength index is expressed as the percentage of material un-degraded in the test and as the percentage of $-\frac{1}{2}$ in. fines generated during the test. The latter is also called *dust index*. The particle sizes chosen as criteria of strength in the method of Voice and associates could be modified to conform with the opinions of our blast furnace people who generally regard material finer than 20 mesh as too fine. This is, however, only a suggestion and some other method might be preferred.

I think that clarification of the issues raised will provide some of the answers needed to establish a good, dependable method for qualitative evaluation of sinter. This method based on the Stephenson and Carney sampling procedure could develop into a standard sinter testing procedure which might be eventually accepted by the whole United States iron ore sintering industry.

⁵ W. O. Philbrook: *Blast Furnace and Steel Plant* (1943) 31.

⁶ F. M. Hamilton and H. F. Ameen: Laboratory Studies on Iron Ore Testing and Sintering. *Trans. AIME* (1950) 187, p. 1275.

⁷ R. E. Powers: *Yearbook AISI* (1951).

⁸ E. W. Voice, C. Lang, and P. K. Gledhill: *Journal of Iron and Steel Institute* (1951) 167.

R. L. Stephenson and D. J. Carney (authors' reply)—

We appreciate Mr. Holowaty's comments on this paper, and we hope we will be able to clarify the points he enumerates. In the following discussion, the individual points are numbered in the same order as in Mr. Holowaty's discussion.

1—Some of the confusion, we believe, has arisen from the fact that only 5-lb samples were used in the preliminary experiments in which the shatter test was compared with the tumbler test, whereas the sample size selected for the final tumbler test was approximately 100 lb. The reason that only a 5-lb sample was used in the preliminary experiments was that separately procured 100-lb basket samples might not be identical and would not constitute suitable starting samples for comparing two degradation tests. To provide samples as nearly identical as possible for comparing the two testing procedures a 500-lb sample was procured from the sinter cooler. This large sample was separated into various size fractions which were in turn riffled into smaller samples. The comparison was run with samples containing pieces of sinter between 1 and 2 in., and also with samples containing pieces of sinter between $\frac{1}{2}$ and $\frac{3}{4}$ in.

Although the drop test was limited to 16 drops, to avoid the excessive amount of work that would be required if a greater number of drops were employed, the effect of using an intermediate number of drops was also investigated. For 4, 8, 12 and 16 drops, the per cent of sinter coarser than each sieve was approximately inversely proportional to the logarithm of the number of drops to which the sinter was subjected.

2—The final selection of the tumbler test, as Mr. Holowaty pointed out, was based to a certain extent on the fact that this test was the most convenient method for doing the job satisfactorily.

3—It is true that initial particle size is an important factor influencing the final size of the particles in almost any type of degradation test, as Mr. Holowaty has stated. However, he must not lose sight of the fact that in our test the original size is uniform, since it is a piece of sinter approximately 1 cu ft in volume. If this piece drops once or twice outside the tumbler drum and then 400 times inside the drum, the final particle size will be for all practical purposes the same as if it had only experienced the 400 drops inside the drum.

4—Mr. Holowaty is right in pointing out that the use of the mean size has serious disadvantages in comparing materials that differ widely in the distribution of the weights of the various size fractions about the mean size. However, in all our experience in tumble testing sinter, we encountered no serious irregularity in particle size distribution, so that the median size of the tumble tested sinter provided a suitable measure for the sinter strength.

F—Coal

Cleaning Various Coals in a Dense-Medium Plant

by M. R. Geer, W. A. Olds, and H. F. Yancey

DISCUSSION

J. S. Huckaba (*Western Machinery Co., Spokane, Wash.*)—It has been my pleasure and privilege to be able to follow this work with the H.M.S. pilot plant very closely. This has been a very thorough and painstaking piece of work and will be of great value to the dense-media operators and manufacturers. Unfortunately time was not available to make an accurate comparison between this pilot plant and the commercial

plant at Bellingham Coal Mines, both plants having Wemco drum separators.

Briefly I would like to point out what I consider the most interesting finding of this work.

Capacity: Very good results were obtained at a feed rate of 5.2 tons per hr in a 30-in. diam by 24-in. long drum separator. The maximum capacity of this drum is not known but it is something over 5.2 tons per hr for the available area of 2.08 sq ft.

Medium Preparation: The importance of proper medium preparation is well demonstrated. A "tailored" or

properly prepared medium will have the required stability and necessary differential without the need for slimes or foreign material to maintain these conditions. With a clean medium and a proper differential an efficiency of 99.9 pct is possible.

Distribution Curve: This curve is often referred to as an "error curve" or a performance curve. It is by far the most useful one available for demonstrating the operating characteristics of any type of separating equipment. Also, it gives an accurate method of determining the specific gravity of separation. I believe everyone dealing with heavy density separation would do well to study this paper, as it contains all the necessary information for plotting this distribution curve. It has been used for some time by the French, but is just now coming into common usage in the United States.

Densimeter: As accurate controls were necessary in so small a unit, a simple, cheap, and accurate specific gravity weighing device was developed. Reprints of the article on mechanization describing this densimeter are available by writing the Bureau of Mines at Seattle.

I do not quite concur with the information given therein regarding the results obtained with contaminated medium as these tests were made at a feed rate far below the demonstrated maximum. If test 43 (with contaminated medium) had been run at a feed rate of 5.2 tons per hr or greater, I am sure the results would have been very much inferior to those made with clean medium. I believe further investigations along this line would be in order before any definite conclusions can be reached.

I wish to thank Dr. Yancey for the opportunity of writing a discussion on this very informative technical paper.

Nelson L. Davis (Nelson L. Davis Co., Chicago)—Mr. Yancey and his associates are to be both complimented and congratulated on the constructive research into the characteristics of dense media coal cleaning as revealed in their paper.

As near as it is possible for anyone to set up small-scale laboratory testing procedure and project the results so that they can be used to anticipate those from commercial size installations, it seems to me that the report is of much practical value.

Certainly the results of this work can be authentically considered to indicate trends. However, as Mr. Yancey has pointed out, the characteristics of coal are widely variable and therefore the results of any small-scale test work are neither conclusive nor specifically dependable.

It will, therefore, be of interest to report a procedure that has been adopted and followed in Europe by the Charbonnages de France. Here seven coal cleaning plants of commercial size were installed for treating coal with dense media at capacities ranging from 100 to 480 metric tph and for feeds ranging in size from 6 to 30 mm to 30 to 250 mm. These incorporated six different types of the best dense media cleaners available. The plants were located in the Pas-de-Calais, Lorraine, Sarre, Central, and Eastern coal fields of France.

The program was initiated in 1949 and completed in 1951. During these two years intensive research and development resulted from the commercial operation of these various types of processes and it was possible to obtain facts which led to the discovery of important characteristics and comparative ratings as to the fitness of each.

From this work, the important characteristics were judged to be:

1—Efficiency with respect to the amount of misplaced float and sink material reporting with the products. In obtaining these results, unit feed capacity was initially low and gradually increased until sampling showed the efficiency of the separation was beginning to diminish.

2—By means of the above described test procedure, optimum unit feed capacities were established for each process being studied.

3—For some of the processes, it was difficult to maintain continuity of accurate density control. Therefore this became the third important characteristic necessary properly to evaluate the various types of processing equipment being tested.

4—Efficient recovery of the magnetite from the rinse slurries was found to be widely variable and, therefore, this was established as the fourth important characteristic.

In conducting this work, it was found that where the size range of the feed was, say 30 to 150 mm, all the above characteristics were better than experienced over a wider size range such as 6 to 150 mm.

During the test procedure, each size of feed was tested with each of the six processes being examined. Out of all this, the following conclusions were gained:

1—All processes showed approximately equal float-sink efficiency for the separation up to the point where optimum feed rate was reached, but such feed rates differed widely for the various processes tested.

2—Categorically, all processes naturally belong to either one of two basic types, namely, top feed vessels where the feed enters at the surface of the bath, or submerged feed vessels where the feed is submerged to a zone considerably below the surface of the bath. Materially higher capacities were gained with the latter type, particularly where more than 10 pct of the feed represented sink having a density not more than 0.10 greater than that of the bath.

3—As the average particle size of the feed material was decreased, there was a corresponding need to decrease the unit feed rate, quite evidently because the rising or settling velocity for small size near gravity material was materially less than for the larger particle sizes.

4—The ability to maintain uniformly constant bath density depended importantly, not only on the efficiency of the recovery for magnetite, but also upon the elapsed time between the point of rinsing the float and sink products and restoration of the recovered magnetite to the bath circuit. Where the elapsed time was least, the control of the bath density was best and easiest from an operating standpoint.

In conclusion, and in view of the strong trend toward the wide use of heavy media circuits for cleaning coal, it is suggested that means be established which could result in owners granting permission to U. S. Bureau of Mines' engineers to enter their heavy media plants and obtain samples of the products regularly being produced, so that in the end the comparative results could be published as case records and be made available to the coal industry. Thereby the industry would gain direct benefit from the basic facts discovered.

M. R. Geer, W. A. Olds, and H. F. Yancey (authors' reply)—We do not agree with Mr. Davis' statement that the results of small-scale test work are neither conclusive nor dependable. Many years of experience have demonstrated that usually laboratory-scale testing is sufficiently accurate for predicting plant performance within the limits required. Moreover, some problems are restricted by their nature to laboratory treatment, while others can be studied more quickly and cheaply in the laboratory than in the plant. Thus the laboratory or pilot plant has a definite place in coal-preparation investigations.

However, some preparation problems can be studied in the plant to much better advantage than in the laboratory; hence Mr. Davis' suggestion that a program similar to that of the French be undertaken in this country deserves serious consideration. The Bureau of Mines has made some plant-performance tests in the past, but never on a scale even approaching that of the work under way in Europe. Perhaps an increase in the tempo of research in this country is necessary, lest our own technology be overshadowed by European advances. The cooperation of coal producers, equipment manufacturers, and research groups in a coordinated, adequately-financed program could pay handsome dividends in the form of improved preparation practice.

The Blending of Western Coals for the Production of Metallurgical Coke

by John D. Price

DISCUSSION

R. W. Campbell (*Jones and Laughlin Steel Corp., Pittsburgh*)—As usual John Price has presented an excellent paper. I know of no one who has devoted more time and conscientious thought to this subject than he has, and his efforts have produced scientific data and conclusions of real value.

Results of similar studies made by the Jones and Laughlin Steel Corp. on the blending of Pennsylvania coals may be of interest.

During 1935 the late F. W. Wagner and the present writer developed an empirical physical fuel value formula for evaluating the combined physical tests of metallurgical coke, see Table IV. In the development of the formula, the tumbler test, shatter test, screen test, and porosity of coke have been taken into consideration. Each of these tests brings out some physical property not expressed by the others, and in estimation of the importance of these tests as a medium for evaluating coke for the blast furnace, their worth has been graded as follows:

Item	Pct
Tumbler, pct on 1 in.	60
Shatter, pct on 2 in.	20
Screen, pct total on 2 in.	10
Porosity, pct	10

It was thought that some additional penalty should be applied to cokes that do not show a uniform screen size, assuming that coke larger than 4 in. was not desirable and that this size was an indication of irregular size.

Therefore the screen test factor is further corrected by applying a penalty of 1 pct for each 1 pct material greater than 4-in. size as shown by the screen test.

The physical fitness of coke for blast furnace use depends upon four fundamental characteristics: 1—general strength, as indicated by the shatter test; 2—abrability, or the resistance of cell walls to reduction in size by attrition, as indicated by the tumbler test; 3—porosity; and 4—size and uniformity. Each of these characteristics is not of equal importance, but surely,

in an attempt to determine an accurate physical value, the individual value of each should be taken into consideration.

The formula is used as a yardstick, so to speak, in determining variations in the normal production of metallurgical coke and in making comparisons between normal coke and experimental blends with other coals.

Table IV gives results of coking tests on raw and washed high volatile coals and these same coals with blends of coke breeze, anthracite coal, char and low volatile coals, all from Pennsylvania. The table also gives the source of the coals and coke breeze, as well as their proximate analyses.

It will be noted that the washing of the raw high volatile coal increased the physical fuel value from 26.17 to 39.16, showing the effect of removing shale and slate which produces a detrimental fracturing and weakening of the coke.

It is a well known fact that many Pennsylvania high volatile coals contain an excess of resinous material or binder which has a tendency to weaken the coke, owing to its high evolution of gases during carbonization. Tests were therefore made with coke breeze, an inert material, in an attempt to alleviate this condition. Blends of washed coal with 5 pct and 7.5 pct coke breeze collected from the coke plant quenching sump (82.9 pct through $\frac{1}{8}$ -in. screen) resulted in cokes with a physical fuel value of 20.75 and 16.73 respectively, lower than results obtained from 100 pct raw high volatile coal. It will be noted that the percentage of + 4-in. coke increased but that the tumbler test was very low. Undoubtedly better results could have been obtained if the coke breeze could have been crushed extremely fine.

During World War II the government asked the coking industry to consider the use of anthracite coal as a substitute for high quality low volatile coal, which was extremely scarce at the time. The next series of tests was made with blends of raw and washed high volatile coals and 5, 10, and 15 pct Pennsylvania No. 5 buckwheat anthracite (100 pct through $\frac{1}{8}$ -in. screen). It was necessary to use some raw high volatile coal in these blends because of insufficient washer capacity at the time. It will be noted that the physical fuel values

Table IV. Physical Coke Tests From Blends of Pennsylvania Coals

Additive →	None	Breeze	Anthracite	Char	Low Volatile
Raw high volatile, pct	100	—	38	34	—
Washed high volatile, pct	—	100	54	51	90
Coke breeze, pct	—	95	7.5	85	80
Anthracite, pct	—	5	—	10	15
Char, pct	—	—	10	—	—
Low volatile, pct	—	—	—	—	5
Coke tests					
Screen, pct on 4 in.	5.8	10.5	10.7	15.5	20.0
Screen, pct total on 2 in.	56.5	72.9	58.0	63.9	80.9
Shatter, pct on 2 in.	30.8	50.5	27.3	23.9	55.2
Tumbler, pct on 1 in.	17.1	29.5	9.1	3.0	25.9
Porosity, pct	44.49	48.22	46.70	47.81	45.66
Physical fuel value	26.17	39.16	20.75	16.73	37.62

Proximate Analyses				Physical Fuel Value			
	V.M.	F.C.	ASH	SUL.	P.F.V. = 0.6(A) + 0.2(B) + 0.1(C) + 0.1[(D) (100-E)]		
Raw high volatile	34.39	55.89	9.72	1.20	Where, A = Tumbler, pct on 1 in.		
Washed high volatile	36.10	57.66	6.24	1.06	B = Shatter, pct on 2 in.		
Coke breeze	3.50	81.50	15.00	1.50	C = Porosity, pct		
Anthracite	4.51	84.85	10.64	0.82	D = Screen, pct total on 2 in.		
Char	15.54	75.55	8.91	1.82	E = Screen, pct on 4 in.		
Low volatile	18.14	74.82	7.04	0.98			

Note: Raw and washed high volatile coal from Pittsburgh seam, Washington County.
Coke breeze from quenching sump, 82.9 pct through $\frac{1}{8}$ -in. screen.
Anthracite, No. 5 buckwheat, 100 pct through $\frac{1}{8}$ -in. screen.
Char produced from Pittsburgh seam coal.
Low volatile coal from Upper Kittanning seam, Somerset County.

were 37.62, 41.94, and 38.77 respectively, and that there was a decided increase in the coke size, namely, the percent on 4 in. and total percent on 2 in., as well as the shatter test. However, there was no improvement in the tumbler test and the coke was much denser, as indicated by the porosity. A comparison of physical fuel values of blends with anthracite, char, and low volatile are as follows:

Additive	5 Pct	10 Pct	15 Pct
Anthracite	37.62	41.94	38.77
Char		47.55	47.89
Low volatile	47.28	47.95	50.00

The gradual depletion of low expanding high quality low volatile coals in Pennsylvania and West Virginia is becoming more apparent each year, and some consideration must be given to a suitable substitute for this coal when necessity demands it. The Pittsburgh Coal Co. produces a char under the trade name of *Disco* from Pittsburgh seam coal (15.54 pct volatile matter), and it was with the thought in mind that some day we might be faced with the problem of producing char as a low volatile substitute from our own high

volatile coals that we obtained sufficient tonnage of this *Disco*, or char, for a series of tests. The next series of tests was made with blends of washed high volatile coal with 10, 15, and 20 pct char. A metallurgical coke of good quality was made with each blend, as the physical fuel values of 47.55, 47.89, and 51.02 indicate. A comparison of the physical fuel values of blends with char and low volatile coal are as follows:

Additive	10 Pct	15 Pct	20 Pct
Char	47.55	47.89	51.02
Low volatile	47.95	50.00	56.59

These tests seem to indicate that to obtain cokes of approximately the same physical fuel values, 5 pct more char would be required in the blend than would be necessary with low volatile coal, for example:

	Char, 15 Pct	Low Volatile, 10 Pct
P.F.V.	47.69	47.95
	Char, 20 Pct	Low Volatile, 15 Pct
P.F.V.	51.02	50.00

Two-Way Belt Conveyor Transportation

by C. W. Thompson

DISCUSSION

Paul D. Suloff (*Goodyear Tire and Rubber Co., Inc., Akron, Ohio*)—I would like first to comment on problems of the conveyor belt discussed in Mr. Thompson's excellent paper, since that is what we hope we know most about. Twists in relatively wide conveyor belt unavoidably produce a lateral maldistribution of tension, raising tension at belt edges and reducing it at the center. They also produce a lateral collapsing force on the belt at the center of the twist owing to the inherent tendency of all the longitudinal elements of the belt to try to pass through a point at the twist center. Calculation of the twist geometry by the methods shown in Mr. Thompson's paper keeps these extraordinary forces within limits which the belt designer can tolerate. No reduction in belt life due to twisting need be contemplated when this geometry is maintained. There is a minor exception that belts of extreme lateral flexibility will tend to curl laterally at the center of the twist. However, any ordinary fabric construction will perform satisfactorily in this respect.

These twists are always made in regions of low tension in the conveyor so that even in the edges of the twist, belt tension does not exceed the average tension found in highly stressed regions of the conveyor.

Offsetting these out-of-ordinary belt stresses is the advantage that Mr. Thompson has brought out of getting the return run up out of the dirt where it can be seen. This not only makes it easier to train, but also, in the event that it is not properly trained, frees it of the normal return run edge wear hazard. It is well known that return run edge wear is a prominent cause of belt mortality underground.

An interesting aspect of this two-way conveyor is that the belt may be made what is known as a *Mobius Strip*. A *Mobius Strip* is obtained by splicing a belt after turning one end of it 180° about its longitudinal axis. In other words, one end is turned upside down before splicing. A belt spliced in this fashion turns itself upside down every time it comes around, but the twist which has been put in the splicing, of course, stays at one location on the conveyor, in this case one

of the twist sections at the end. Turning the belt over every revolution might have advantages in some cases. Belts could be made with equal covers and the two sides worn equally and simultaneously. In this case there would be no problem of getting belts on upside down by mistake. However, the two-way conveyor does not have to be a *Mobius Strip*. It can be twisted in such fashion that the same side is up on both runs. It is simply a question of which way the final 90° twist is made before joining the ends.

Another interesting aspect of the two-way conveyor is the problem of operating two-way conveyors in series. Here the sequencing of starting brings up some new problems. It will be recognized, although not always at first glance, that if the starting sequence is planned for one run of the conveyor the reverse will result on the other run. With the two runs carrying bulk material in both directions a reverse sequence on one run would be intolerable. In this situation the only solution appears to be a simultaneous starting of all conveyors in the series. However, with the coal in one direction and intermittent supplies in the other it would be entirely practical to sequence the conveyors for the coal run and accept a reverse sequence on the supply run.

The two-way conveyor also lends itself to new driving possibilities. First, it is quite possible to drive at the head end of each run, which of course, means a drive at each end of the two-way conveyor. Driving in this way a given belt can be extended to substantially greater lengths than a conventional conveyor with drive at one end only. In addition to this, under certain conditions the conveyor can be extended to extreme length by driving at one end and at some intermediate point on the most heavily loaded run. As a particular case, a belt carrying coal downgrade and supplies back upgrade could be extended to extreme lengths by driving at the head of the coal run and at an intermediate point of the supply run.

Mr. Thompson has been a pioneer in belt conveyor transportation underground and his accomplishment here with the first two-way conveyor of any consequence is another notable addition to the art.

H — Industrial Minerals

Titanium Dioxide Analysis of MacIntyre Ore by Specific Gravity

by Alan Stanley

DISCUSSION

R. T. Hukki (*Finland Institute of Technology, Helsinki, Finland*)—In the analysis of mill products containing minerals of appreciable difference in specific gravity the well-known principle of the method described by Stanley is applied in a number of ways for a variety of problems. The use of Le Chatelier bottle as explained in the paper may be a dependable and sufficiently accurate way, although no information is given as to the reproducibility of the specific gravity determinations. In the case of MacIntyre ore the method is applied for three separate products produced apparently without reagents and relatively free of fine slimes. Should it be applied for flotation products carrying a substantial amount of slimes, difficulties should be anticipated in the process of eliminating air inclusions from the sample.

The difficulties mentioned above may be overcome by the use of a volumeter, an instrument designed for specific gravity determinations of finely divided solids in dry state. This instrument, see Fig. 4, is manufactured by F. Süssert-Bürner A. G. in Basel, Switzerland. Its operation is based on the law of Boyle and Mariotte.

With the use of the volumeter, the specific gravities of test samples having a total volume of about 30 cm³ are easily reproducible within ± 2 units in the second decimal of the specific gravity figure, and even better than that with utmost care. This accuracy is sufficient for ordinary operational control. How closely the specific gravity determinations follow the respective chemical assays depends primarily on the nature of the sample itself. In this respect no specific gravity instrument can give better results than those justified by the consistency of mineral distribution in samples analyzed.

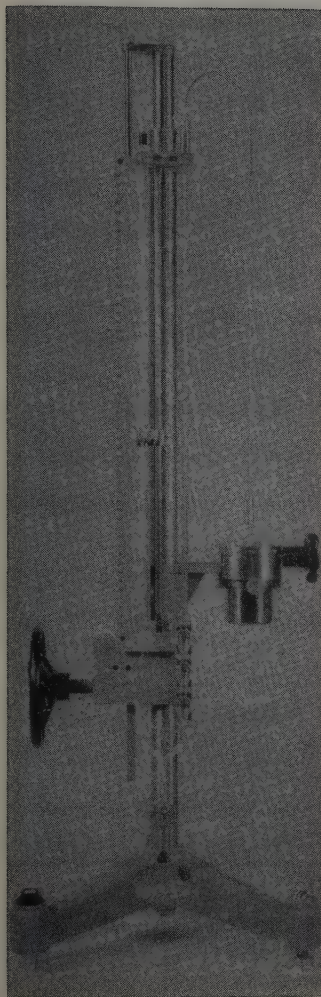


Fig. 4—The volumeter shown here is manufactured by F. Süssert-Bürner A.G. in Basel, Switzerland. By means of this instrument specific gravities of test samples having a total volume of about 30 cm³ are reproducible within ± 2 units in the second decimal of the specific gravity figure.

I — Mining Geology

The Role of Geologists in the Development of The Labrador-Quebec Iron Ore Districts

by J. K. Gustafson and A. E. Moss

DISCUSSION

A. K. Snelgrove (*Michigan College of Mining and Technology, Houghton, Mich.*)—To one who was officially associated with the granting of the Newfoundland Labrador concession this paper is of special interest. In congratulating the authors, who have placed on record a valuable case history of accomplishment by the concerted effort of a large number of geologists and others, I should like to add a few side lights.

The original concession of over 20,000 square miles was negotiated in 1935 with the Newfoundland Commission of Government by J. H. Colville of Toronto for Weaver (Minerals) Ltd. This company was succeeded by the Labrador Mining and Exploration Co., Ltd., in 1936. The Geological Survey of Newfoundland, after a study of regulations governing mineral conces-

sions throughout the British Empire, urged upon the Commission of Government that a prime condition of the concession agreement should be the conduct of the geological exploration at a scientific level comparable to that maintained by the Geological Survey of Canada under similar circumstances of accessibility. The Newfoundland Survey, then being revived, was preoccupied with an extensive program of areal and commodity surveys in the island itself; the Survey welcomed the yearly increment to the geological knowledge of a hitherto practically unknown one-fifth part of the mainland dependency as represented by the required reports of the company's operations. Naturally these reports could not be published during the selection-rejection period. Dr. Retty's achievements from these early days onward have been appropriately signaled

in the award to him of the Selwyn G. Blaylock medal by the Canadian Institute of Mining and Metallurgy in 1952.

I was privileged to inspect the progress of the field work by visits in the summers of 1937 and 1946. In talks to four AIME sections, accompanied by movies taken on these tours, I was sometimes confronted by scepticism about the possibilities of this remote iron district. It is reassuring to witness iron-ore production in sight from this new major field.

The Labrador project now ranks with the central Africa copper explorations in the employment of geologists. The contravention of a rule in the cycle of metal production that in a new territory a commodity of low unit value is the first fruit (e.g. Labrador's counterpart, Alaska) is attributable not only to geological vagaries but to strong capital in addition to the geological organization described in this article. Of course, the Stag Bay placer and Wabush Lake hard rock gold rushes of 1923 and 1933, respectively, stimu-

lated interest in Labrador minerals generally, and while those incidents are best forgotten, the non-ferrous potentialities of Newfoundland Labrador, an eighteenth part of the Canadian Shield, will now be more adequately assayed with the introduction of railway transportation.

The relative effectiveness of the professional prospector and of the geologist, with their differing immediate concerns, in the discovery of mineral deposits is ever a lively topic. It would be of interest if the authors were to enumerate which of the 44 known deposits can be credited to the college boys. Furthermore, the Labrador-Quebec district since 1936 has been a valuable training ground for young geologists, and I think another significant part of the record would be the number of graduate theses which have been based on this fascinating episode of mineral exploration.

The promised detailed geological report of the staff of the Iron Ore Company of Canada is anticipated with keen interest.

L — Geophysics

Geophysical Case History, Fredericktown Lead District, Missouri

by Harold Powers, LeRoy Scharon, and Carl Tolman

DISCUSSION

Sherwin F. Kelly (*Pres., Sherwin F. Kelly Geophysical Services, Inc., Amawalk, N. Y.*)—The authors of this paper have demonstrated in an effective manner a kind of application of geophysics which is probably destined to play a role of ever-increasing importance in future prospecting campaigns. The techniques utilized for the direct detection of mineralized bodies suffer from various limitations, and where such limitations prevent successful application, the indirect attack, as exemplified in the present survey, will often prove useful. The indirect attack requires a very careful correlation of geophysical and geological data and therefore necessitates a thorough cooperation between the geologist familiar with the area and the geophysicist called in to assist in the prospecting campaign.

The authors refer to the earlier spontaneous polarization survey carried out by my firm, and a word of clarification is in order. The spontaneous polarization technique evidently did not prove as useful in tracing extensions of known ore bodies as had at first been hoped, but in fairness to the method employed attention should be drawn to qualifying statements embodied in the recommendations we submitted at that time. Statements were specifically made to the effect that individual areas of strong electrical activity *might* or *might not* correspond to commercial ore; they *would* correspond to stronger sulphide mineralization, but this would not necessarily mean the presence of ore grade. It was warned that occasional disappointment must be anticipated when drilling such areas of strong electrical activity. On this account, we believed that the most effective role for the spontaneous polarization technique would be that of searching for new mineralized areas in virgin territory. By utilization of the method on a reconnaissance basis in new territory, hitherto unsuspected mineralized zones, similar to that near Fredericktown, could easily be discovered, if they exist. I should like to emphasize this role for the spontaneous polarization method, because at the time of the Fredericktown survey we believed that this would be more useful than attempting to utilize it for locating extensions of known deposits. We suggested that such reconnaissance by the spontaneous polariza-

tion method should be conducted in conjunction with a magnetic survey for the purpose of tracing the buried porphyry ridges. Until a program such as we recommended is put into operation, it is premature to express a final conclusion as to the utility of the spontaneous polarization technique in prospecting for mineral deposits of the Fredericktown type.

The resistivity technique employed by the authors of this paper can no more indicate the presence of *ore* than can the spontaneous polarization method. Therefore any indications of sulphides which are obtained in the course of resistivity depth determinations are subject to exactly the same doubts and uncertainties as are the strong potentials, indicative of sulphides, obtained in the course of a spontaneous polarization survey. The authors admit that the method does not reveal galena mineralization directly, and I would therefore take exception to the statement in the first sentence of the next to the last paragraph, that the electrical resistivity method "... established the limits of the orebody south of shaft No. 1 ..." No geophysical method now in use is capable of delimiting ore, *per se*. It can only establish the limits of sulphide mineralization.

It seems to me that an effective procedure for a reconnaissance survey to search for additional mineralized districts, such as that at Fredericktown, would be to apply the spontaneous polarization method first on a wide grid of observations. This would reveal any new and unsuspected zones of sulphide deposition. Such reconnaissance should then be followed by magnetic work in the same area, to determine the pattern of porphyry ridges, knobs, and troughs. Within the areas thus delimited, electrical resistivity work both by traverses as well as by depth determinations, as used by the authors, should then be conducted for obtaining more precise data relative to the porphyry-sediment contact. A detailed spontaneous polarization survey could then be utilized to reveal the zones of maximum sulphide mineralization and thereby assist in completing the picture of sulphide occurrences. By such a procedure, the most effective guidance could be given to a drilling campaign to determine where sulphides of ore grade could be anticipated.

The Scintillation Counter in the Search for Oil

by R. W. Pringle, K. I. Roulston, G. W. Brownell, and H. T. F. Lundberg

The rapid improvement of the airborne scintillometer and the perfection of its efficiency for counting low energy gamma radiation has made it possible to work out a technique to map in great detail the radiation pattern at the earth's surface. On such maps low radiation over certain areas appears to indicate the existence of oil accumulations, forming a pattern similar to that obtained by the geochemists.

RADIOACTIVE analyses of samples from the surface of oil fields were carried out more than 10 years ago in Alberta by the alpha particle ionization chamber technique,¹ but large enough tracts could not be covered in these investigations to make possible any evaluation of the method as a means of oil exploration. Considerable interest has recently been revived, however, as a result of certain striking advances which have been made in the instrumentation available for the measurement of radioactivity. It is the object of this paper to indicate the nature of these improvements in radiation technology and then to describe the attempts that have been made to interpret the radioactive patterns obtained in the course of airborne recordings with the new instruments. Since the survey can be carried out from the air and records can be accumulated over vast areas in a short time, the result may easily lend itself to statistical treatment. Areas have been surveyed in Alberta, British Columbia, Saskatchewan, Quebec, Texas, New Mexico, Nebraska, Colorado, Utah, and Montana. Producing fields in Alberta and West Texas have been flown over several times in different directions, Fig. 1. The operations were then extended into unknown territory and drill holes were put down on the anomalies which looked promising. The results from these drillings were encouraging and have given hopes for the development of an entirely new method of oil exploration.

Any large scale method for the survey of radioactive anomalies must be based on the measurement of gamma rays, as beta and alpha rays have much too short a range to be of any significance. Thus the essential improvement which has made the present stage of this work attainable is the development of new highly sensitive detectors for gamma radiation. In the past the only detectors of any consequence that were available were the ionization chamber and the geiger counter, but both of these suffer from the defect that only a small proportion of the gamma rays passing through the counter are detected, possibly 0.1 to 0.2 pct. The recent development of the scintillation counter^{2, 3} has completely transformed the situation and has had a considerable impact on many branches of nuclear technology.

The detection of alpha particles in zinc sulphide screens by visual observation of the individual

scintillations which these particles produce dates back to the early spinthariscopes of Rutherford and Crookes, but the combined use of an appropriate scintillating phosphor and photomultiplier tube had to await the technical development of the latter many years later. With this development came the modern era of the scintillation counter and a knowledge of phosphors which have a large light output under the bombarding action of gamma radiation. Some of these phosphors are relatively dense and are capable of stopping a large proportion of the incident gamma radiation. As the sensitive region is the whole volume of the crystal, a very high detection efficiency, 50 pct or more, can be obtained for medium energy gamma rays.

Scintillation counters for geological purposes were first developed in 1949⁴⁻⁶ in an attempt to utilize this remarkable improvement in efficiency, which has the attractive consequence that only a small portion of the normal background of the counter is due to cosmic radiation. In 1949 tests were made in northern Saskatchewan by Lundberg Explorations Ltd. with portable scintillation counters which gave excellent results in the search for uranium and served to indicate unknown uranium deposits in areas previously closely surveyed with geiger counters. Portable scintillometers (registered in Canada) are now commercially available and in regular use,⁷ and the adaptation of the instrument to radioactivity oil well logging has also been very successful.⁸ Initial attempts to measure radioactivity from aircraft with scintillation counters were made during this period in the same area and yielded most encouraging results. It would be appropriate to consider some specific requirements for airborne investigations.

The essential problem to be met in the detection of any radioactive source is the necessity of obtaining a signal greater than the statistical fluctuations of the background counting rate for the instrument. It is possible to show that $Nt > 2k^2$ is the condition for detectability of a signal where

N = average background counting rate for the detection.

t = time constant of the counting rate meter, used to determine the average number of counts arriving in a certain predetermined time interval.

N' = average source counting rate at the detector.

k = N/N' , and $N \gg N'$.

Sample values are given in Table I.

Assume that the aircraft carrying the equipment is travelling at 120 mph, in which case it will cover 176 ft in 1 sec. Assume also, as a first approximation, that a point source target is in range when the air-

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Discussion on this paper, TP 3642L, may be sent (2 copies) to AIME before Feb. 28, 1954.

Manuscript, Aug. 21, 1952. Revised July 6, 1953. New York Meeting, February 1952.

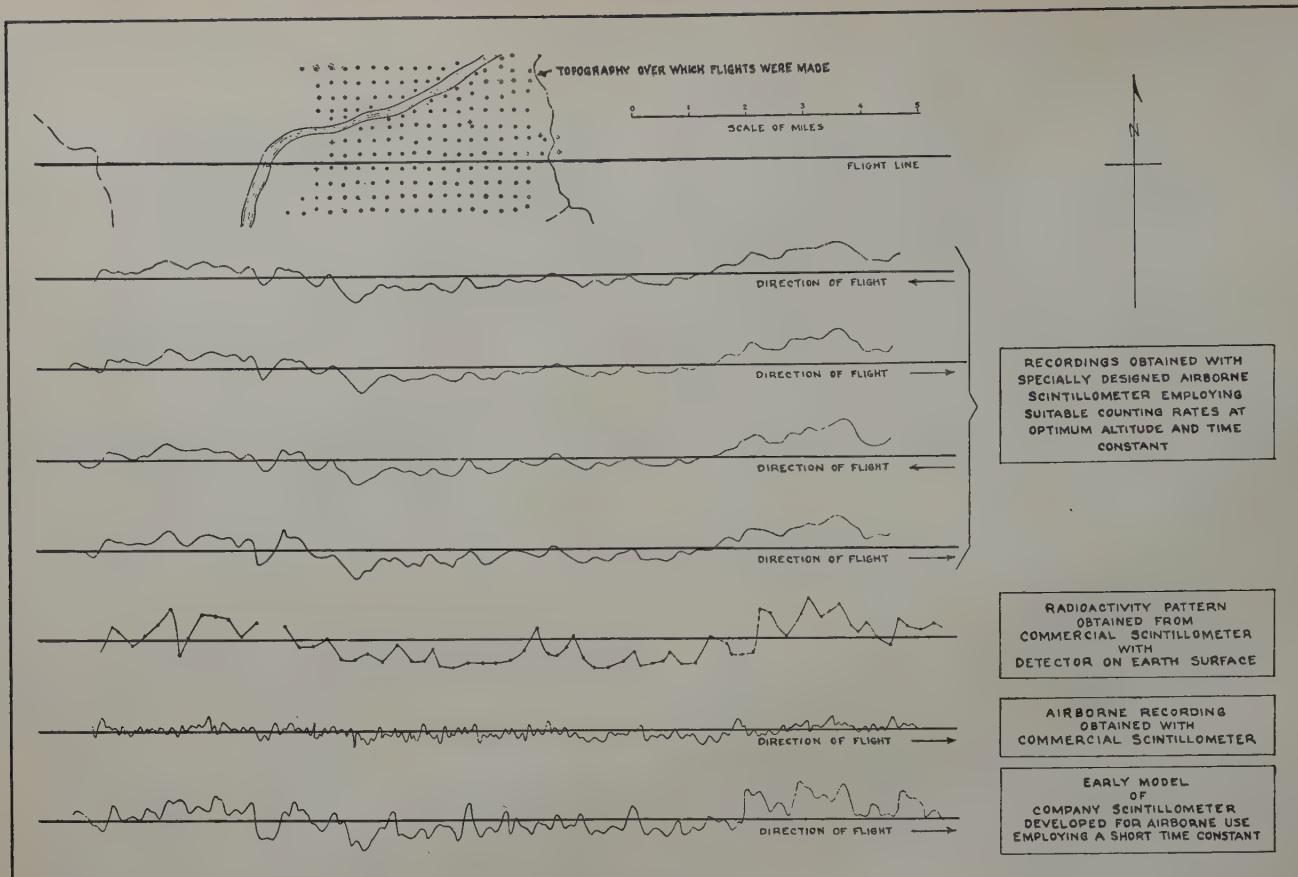


Fig. 1—Flight line repetition examples.

craft is within about 500 ft of the source. Beyond this, radiation is slight, owing to combined effects of atmospheric absorption and the inverse square law for radiation. This corresponds to 400 ft on each side of the source in a horizontal direction if the aircraft height is 300 ft, or 3.5 sec of flight time. To insure that scintillometer response may be sufficiently rapid to give a true indication of the anomaly it is evident that time constants not longer than 1 sec must be used. Thus values are obtained for N as given in the last column of Table I. These extremely high rates for the ordinary gamma ray background in an average locality can be achieved by 1—using large detecting crystals, 2—using several crystals and adding the counts from each one,

3—using special low noise multipliers to enable more of the smaller pulses to be counted.

In addition the situation can be somewhat improved by the introduction of gamma ray shielding to reduce the background counting rate without materially affecting the signal from the ground. This shielding will also minimize the effects of gamma radiation from radon in the atmosphere and

Table I. Sample Values for a Radioactive Source Detected by the Scintillometer

Signal Just Detectable, Measured as Percentage Increase Over Background	k	Nt	Value of N , Counts Per Second for t of 1 Sec
2	50	5000	5000
5	20	800	800
10	10	200	200
20	5	50	50

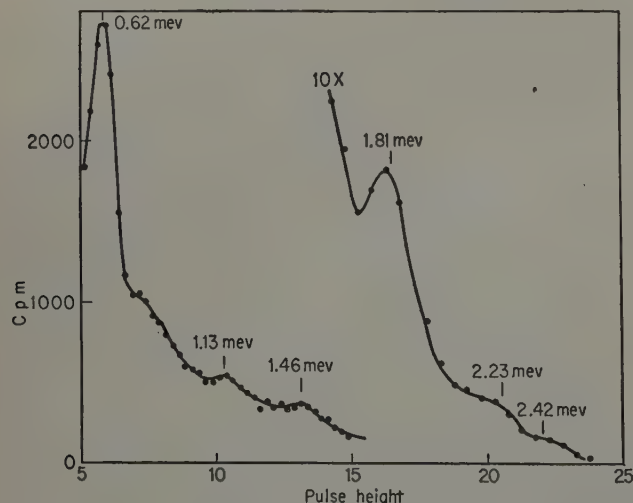


Fig. 2—Ra gamma ray scintillation spectrum.

from unwanted areas on the ground. It might be noted at this point that whereas the radon contribution to the average total ionization, including alpha, beta, and gamma rays, at 300 ft in the atmosphere, might be quite considerable in certain localities, the scintillation counter as used here is measuring only effects due to gamma radiation, so the influence of atmospheric radon is much smaller than it might appear at first sight.

It is possible to translate the above figures in terms of the practical results achieved. It has been shown that it is possible to regard a radiation intensity of 5×10^{-11} rontgens per sec as just detectable in the average locality, with a time constant of 1 sec. This amount of radiation corresponds roughly to the gamma radiation from 1 mg of radium, or 25 lb of

uranium ore assumed at 30 pct U_3O_8 at a distance of 200 ft. A much less sensitive detector might achieve the same result but would require a correspondingly longer integrating time and therefore be of no practical value in aerial survey work. The importance to be placed in the accurate delineation of very small percentage anomalies from the air will be indicated below, but before any discussion of the nature and origin of these very small signals it is opportune to mention another property of the scintillation counter that is of great significance in this problem.

The scintillation counter has the desirable property of acting as a proportional device, so that in essence it is a gamma ray spectrometer⁹⁻¹¹ in which a measure of the energies of the gamma rays is obtained from the pulse height distribution of the voltage pulses from the photomultiplier. The assumption is made, of course, that there exists a strict proportionality between the total light output per scintillation in the phosphor and the energy imparted by the gamma ray to the crystal, an assumption that has now been amply verified. The interaction between the gamma ray and the crystal may take place according to three processes well known to the nuclear physicist: the photoelectric effect, the Compton effect, and the pair production effect. The end result of this is that it is therefore possible to identify in the voltage pulse height distribution curve for a given gamma ray energy, either a photoelectron line, or a pair production line, or a distribution corresponding to the Compton effect, or a combination of these. Fig. 2 shows the pulse-height distribution obtained with the gamma rays from radium using a NaI (Tl) crystal, and Fig. 3 is for the gamma rays of thorium. Considerable attention has been given to the development of methods for the rapid analysis of distributions of this type, using single channel differential discriminators, multi-channel kicksorters, and photographic storage methods employing a cathode ray tube. The latter group of techniques is the most suitable for field use. The value of this proportional property of the scintillometer in geographical investigations lies in the ability of the device to identify a particular anomaly in terms of the spectrum, as well as the intensity of the gamma radiation involved, and thus indicate the nature of the radioactive atoms responsible. It might be noted that the gamma radiation from potassium consists of a single line at 1.47 mev. Fig. 4 gives the dependence on energy of the spectrometer resolution R , defined as the full width of a line at half height.

Oil Field Surveys

To obtain the radioactive data over an oil field, continuous gamma ray recordings are made along flight lines flown over large areas following a predetermined pattern. The flight altitude may vary between 150 and 500 ft above the ground. The speed of the aircraft is not important, although 100 to 150 miles per hr has been found to be very satisfactory. The records obtained are processed and adjusted as to linear scale and then plotted directly along the flight lines on the maps. The results of flights over known oil fields and areas of different types of topography and drainage pattern indicate clearly that great care must be taken in making the interpretation. It is found, however, that radioactive lows generally are obtained over oil fields and that these lows are commonly surrounded by radioactivity slightly higher than normal. The variations in intensity may look erratic over broken topography,

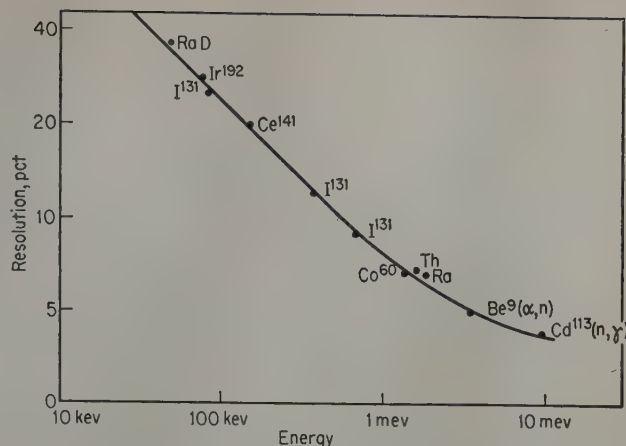


Fig. 3—Th gamma ray scintillation spectrum.

lakes, swamps, and river valleys, but as a rule lows above the oil fields can still be observed. In areas where radioactive shales occur near the surface high intensities may be noted. Lakes, rivers, and sometimes areas of swampy land show marked lows.

On Figs. 5 and 6 are shown radiation profiles along flight lines 1 mile apart over Redwater oil field in Alberta. East-west lines are shown on Fig. 5 and north-south lines on Fig. 6 of the same area. Fig. 7 shows radiation profiles along east-west flight lines ½ mile apart over Coalinga oil field in California.

An arbitrary datum of 150 counts per sec has been used. Radiation intensity higher than normal is shown above the datum line, while lower intensity falls below the datum.

The masking effect of surface water in swamps and rivers may be noted but, in spite of this, most of the area now known as oil producing shows low intensity.

Radioactivity surveys, as made with an airborne scintillation counter, contribute information that is most effectively employed in conjunction with other types of geophysical data. Consider a large tract of territory covered by airborne electric or magnetic surveys or by ground seismic surveys within which several anomalies indicative of structural highs occur. Should the radioactivity survey with its variable pattern over this same area show a radioactive low coinciding with one of the magnetometer anomalies, the probability of an oil pool at this particular site is greatly strengthened. As for every decision to drill, all data must first be assembled, and the radioactivity survey is a contribution to this material.

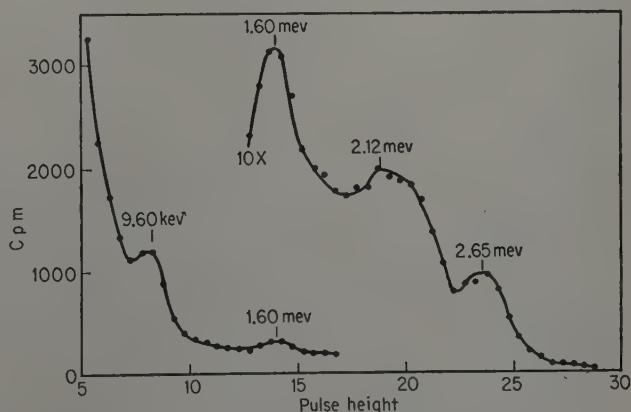


Fig. 4—Resolution energy relationship.

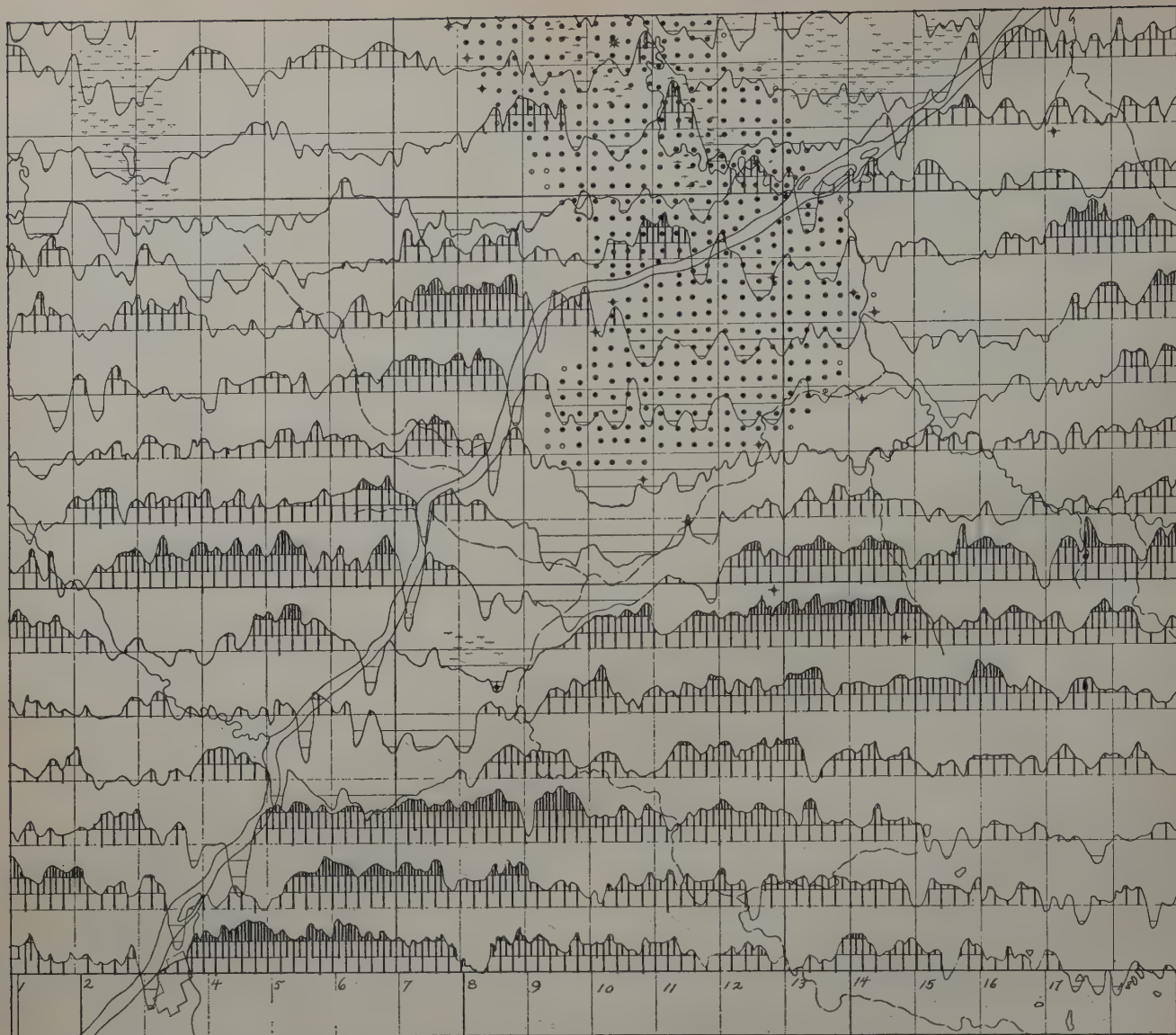


Fig. 5—Radiation profiles along east-west flight lines 1 mile apart over Redwater oil field in Alberta. Intensity scale 1 in. to approximately 100 counts, with arbitrary datum at 150 counts per sec.

It is thought that results to date have justified the incorporation of the scintillation counter among the tools used for oil exploration. But to carry out interpretations to the best advantage, it is necessary to find a logical explanation of the distribution of the radioactive elements in the soils above an oil producing field and in its vicinity. Answers to the following questions must be found: 1—What is the radioactive substance? 2—Where does it come from? 3—How does it get to the surface? 4—What is the explanation of the low radioactivity over oil fields and the increase over the edges?

The uranium-radium disintegration series must be considered before the first question can be answered. Radium is constantly being formed from uranium, and its salts have a water solubility suggesting that the radium stage of the series is the more important in the upward transportation of the radioactive materials. The gamma radiation detected in the surveys comes mostly from the daughter products of radium, of which RaC is the most important. Because of the very short life of RaC, the supply of this isotope must be continually renewed. Since the half-life of radium is 1600 years, it would not be necessary to have uranium itself transported to the surface layers. It is necessary,

however, to have a source of uranium in the formations below the surface and also necessary to have a process of solution, transportation, and precipitation in continuous operation to bring the radium, formed from the disintegration of the uranium, to the surface where it may be sufficiently concentrated to emit gamma radiations of intensities strong enough to be detected by the airborne instruments.

It has been demonstrated that some shales carry an appreciable amount of uranium. Many years ago Russian scientists inspired by Nordenskiöld's discovery in 1893 of a high uranium content in some Swedish oil shales, found that oil field waters could be high in radium and that this radium could be precipitated by adding sulphates to the water. In fact a small quantity of radium was produced in this way at Uhta Oil Field in north Russia. Uranium apparently is present in any major marine sedimentary column, especially in the shales. But it remains to find out how the radium is brought so close to the surface.

Before the second, third, and fourth questions are answered, a study should be made of some of the few geochemical maps and results that have been published. It appears from the maps that hydrocarbons reach the surface by leakage from the oil

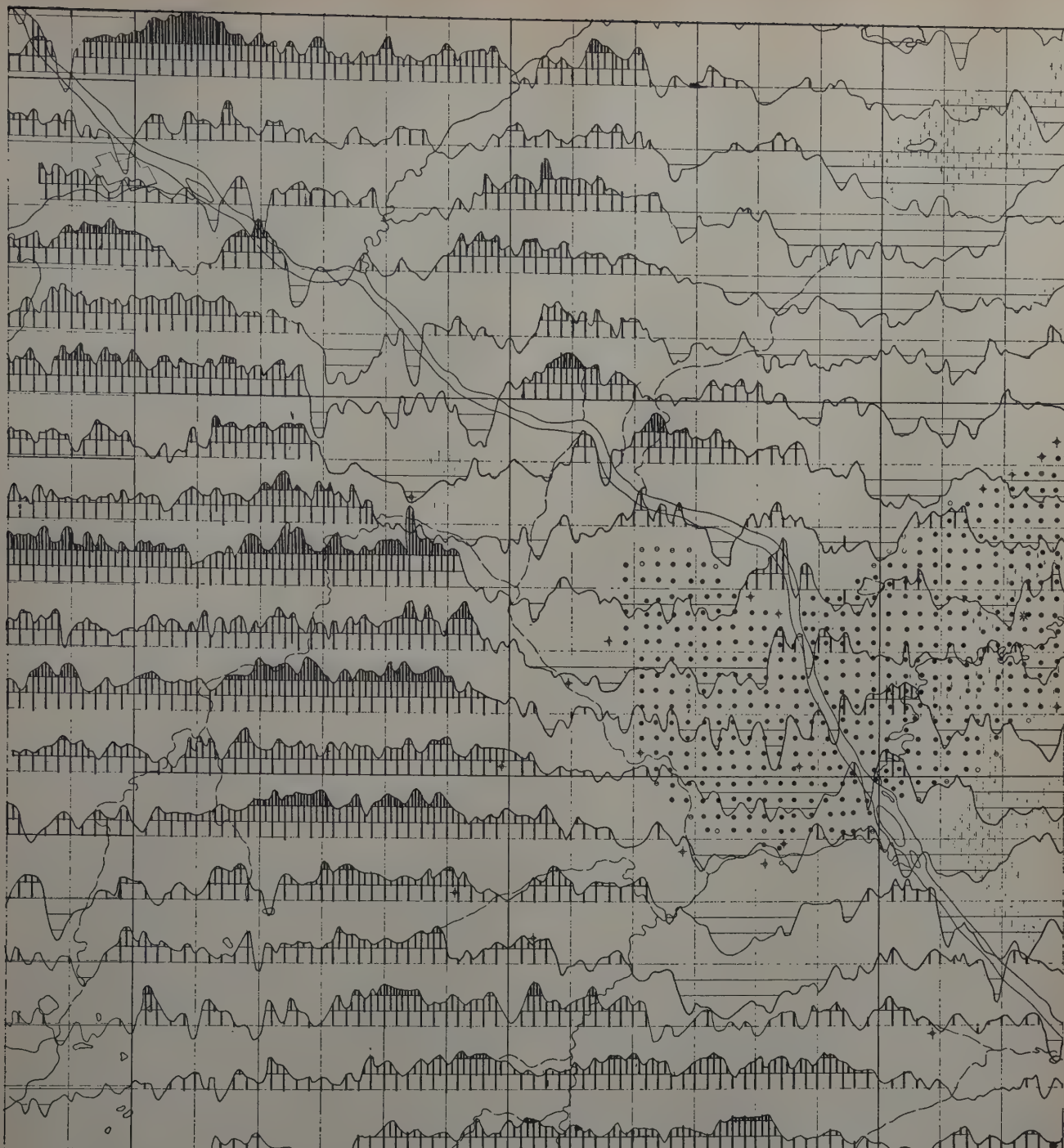


Fig. 6—Radiation profiles along north-south flight lines 1 mile apart over Redwater oil field in Alberta. Intensity scale 1 in. to approximately 100 counts, with arbitrary datum at 150 counts per sec.

deposit. The maps often show a hydrocarbon concentration as a ring or halo around the oil field, sometimes also accompanied by a concentration of chloride. This would mean that beds, against which the hydrocarbons are accumulating, never become completely impermeable until the oil has replaced the water. It also shows that salt water, as well as the hydrocarbons, can travel upwards more or less vertically to the surface. If an oil accumulation is in the way of the upward-trending salt water, it is forced to take the way around the edges of the oil, which means that the speed of the current around the edges is increased. Over the oil pool, on the other hand, the flow should be very slow. This may explain the ring or halo of high hydrocarbon or chloride concentration observed at the surface, as well as the minimum right above it. However, in this connection opinions are still at variance.¹²

The conditions described would favor the forming of similar anomalies in the radioactive radiation from the surface. If hydrocarbons and salt water move together towards the surface, any sulphate contained in the water is reduced to sulphide by the hydrocarbons present. Such a reduction of the sulphate in the surface waters will also take place if minute oil drops or gas bubbles should find their way to the surface. Since the atmospheric ground waters are rich in oxygen, they generally also contain sulphate. A reduction will also take place if the atmospheric waters mix with waters from below carrying some dissolved oil. It can be shown that hydrocarbons are capable of reducing a great deal of sulphate. Even a small amount of oil that may be dissolved in water (approximately 0.02 pct) is sufficient to reduce completely a saturated solution of

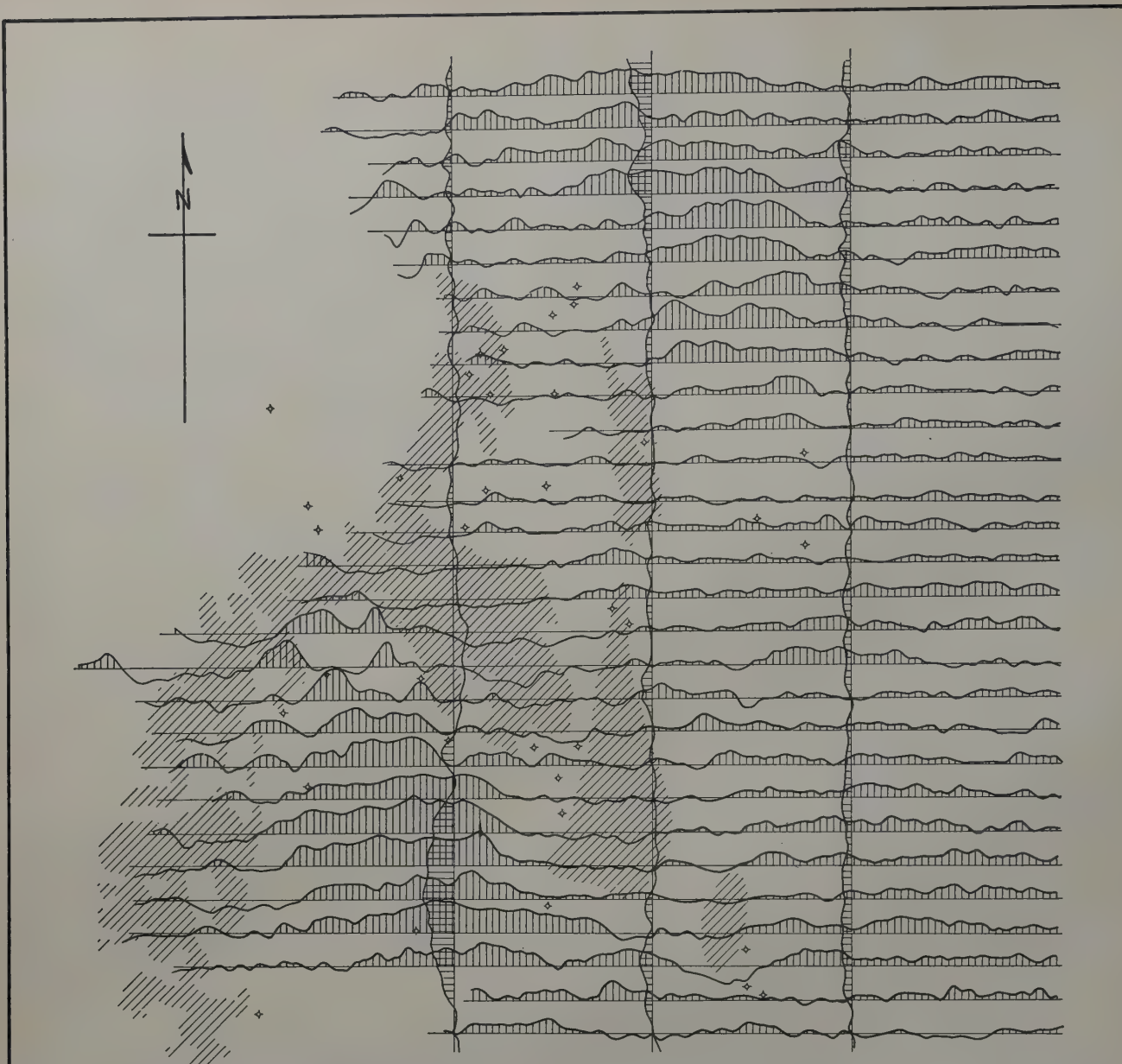


Fig. 7—Radiation profiles along east-west flight lines $\frac{1}{2}$ mile apart over Coalinga oil field in California. Recordings obtained with specially designed airborne scintillometer employing optimum counting rate at optimum altitude and time constant. Horizontal scale 1 in. to the mile. Intensity scale approximately 200 counts per sec.

calcium sulphate. Ordinarily the ground water content is much lower.

Sulphate-free waters may dissolve radium as it is formed from uranium in the rocks, and this would explain the sometimes exceptionally high radium content carried in water from below. Even if the uranium content in these waters is high, as compared with other waters or the oil, this uranium content is very low compared with the uranium content in the ground. It must, therefore, be assumed that the radium content in the sulphate-free waters originates from leaching of uranium minerals and not from any original uranium content in the water or in the oil.

This would mean that a minute leakage of hydrocarbons, sufficient to reduce all sulphate in the ground water to sulphide, would increase the radioactivity of the water considerably without the waters having been in contact with rocks especially high in uranium, such as the black shales which are sometimes assumed to be the origin of the oil.

It is rather unlikely in fact that radium which originates from deep-seated, uranium-bearing rocks or formations close to the oil deposits is the cause of the radioactive anomalies at the surface, since the bulk of the radioactive substance which may be transported to the surface from such great depths would disintegrate and lose its radioactivity on the way up. Even with as fast a movement of the water as 1 in. per year, less than 2 pct of the original radioactivity would remain after rising 800 ft. Therefore the anomalies in the radiation, which have been observed over oil fields, could not be caused by any upward transportation to the surface of radioactive substances from the deep-seated levels of the oil concentrations. If any relation exists, it probably would be found in the fact that the hydrocarbon leakage to the surface has given to the waters at relatively shallow depths the power to dissolve radium from the surrounding rocks. This water, being on its way upward, presses towards the surface with varying velocity, the greatest velocity

being at or near the edges of the oil deposit. This is in agreement with what the geochemists claim to have found, namely, that a hydrocarbon leakage exists together with the transportation of relatively chloride-rich waters towards the surface near the edges of an oil deposit. Even the most penetrating radioactive radiation is stopped by a few feet of rock or soil. Therefore, the radioactive substance must be transported to the surface so that the radiation may be observed from a plane. This transportation of radium to the very surface might be objected to, since on approaching the surface the upward moving waters would encounter air, and the oxygen would then turn the sulphides to sulphates. It is known, however, that in the presence of organic matter, the free oxygen and even the sulphates may disappear at surprisingly shallow depths. In salt marshes the anaerobic activities are known to begin less than 1 in. below the surface.

An investigation of ground waters was made for a period of several years in Sweden at the Lanna Experimental Farm,¹⁸ where certain very important data were gathered. During the winter the sulphate content was at its maximum, 28.8 ppm, but this content diminished during the summer and the water during the late summer became free of sulphate. At the same time the concentration of dissolved solids reached a maximum which indicated evaporation and upward movement of the ground water even close to the very surface. The ground water movement was reversed to downward in the fall and winter, and then the sulphate content reappeared. The amount of organic matter consumed to reduce the sulphates corresponded to only 1/40 of the quantity of oil that may be dissolved in water, or to a daily leakage per square meter of 2 mg of oil or 2 cu cm of methane, i.e., 1 gal of oil or 100 cu ft of methane per acre per year. From this it becomes apparent that even a microscopic hydrocarbon leakage from an oil or gas deposit could make the ground water, at least sometime during the year, free of sulphate all the way to the surface. If then, as at Lanna, the water moves upwards through evaporation, any dissolved radium would be transported to the surface. The change takes place and the superficial ground water receives oxygen and at the same time begins to move downwards. The radium previously transported up will not necessarily again follow the ground water downwards. On the contrary, it is more likely that most of the radium is precipitated in the ground as sulphate, will remain near the surface, and when the sulphates again disappear will get another lift towards the surface.

It has been assumed that radiation from the ground, as observed and recorded from aircraft, must come mainly from Radium C, which is a product of radium and radon. Radon is a gas and can move closer to the surface than radium and naturally diffuses into the free air. Therefore, it would seem as if there would be a good chance, in the vicinity of oil or gas deposits, for a natural transport of radioactive substances to the surface of the ground so that its radiation could be measured and recorded from a plane if it does not fly too high. Tests across Redwater field, Alberta, in August 1951, verified this point. Soil samples were taken at depths of 1, 3, and 5 ft from a series of test holes across the field. Gamma counts were also taken on the surface of the ground at each location. The gamma counts on the soil samples as plotted against geographical position corresponded not only to the

surface count, but also to the continuous airborne recordings.

A gamma ray spectrum analysis as described in the previous section was performed on the soil samples. Those from the oil field perimeter, where the counting rate was high, gave evidence for the presence of Ra and its decay products, whereas in the regions of low counting rate over the oil field the spectrum appeared to be largely potassium, K⁴⁰.

The radiation is obstructed by the water in lakes and rivers. Marshes and swamps may also blot out the radiation. However, in some cases, a vertical transport of radioactive substances through the peat, but naturally not through moveable water, seems to take place, so that it is possible to obtain variations from the bottom of the peat layers, peat being a strong reducing agent. However, it is necessary to make corrections for the lowered radiation intensity before any anomalies can be compared with those over drier soil.

It may be held as an objection to this theory that shallow waters containing oxygen and sulphate and having a downward and lateral movement should prevent the forming of any geochemical anomalies at the surface that could have any relation to the underlying, deep-seated oil or gas deposits. Such conditions may not be uncommon, but are they so frequently prevailing that the method becomes unworkable? In this connection it is necessary to bear in mind that a very great portion of the earth's oil and gas resources have been found by drilling near oil or gas seepages which could have been noticed by anyone. It is probable that there must exist a great many more deposits that only reveal their presence by gas or oil traces which are difficult to detect at surface. It may be mentioned here that in some cases radiation *high*s were noted over certain fields in production, particularly over older fields in Texas. This may be explained from the fact that in some oil fields, where in the beginning no one was particular regarding the disposal of brines, such areas could have been radioactively contaminated. The value of the radioactive prospecting method can be determined only after a sufficient number of newly found anomalies have been drilled and a percentage of successes can be compared with that of already existing methods.

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The Coal Industry in Northern Wyoming and the State of Montana

by Walter J. Johnson

The coals in northern Wyoming and Montana are free-burning and non-caking and range from lignite to bituminous C in rank. Strip and underground mining are employed to supply railroad, utility, industrial, and domestic fuel. The market area includes Iowa, Nebraska, Minnesota, North and South Dakota, Wyoming, Montana, the Pacific Northwest, and western Canada.

THE larger producing coal mines in northern Wyoming are located in the Powder River Basin lying between the Black Hills region of South Dakota, eastern Wyoming, and the Big Horn Mountains in north central Wyoming, see Fig. 1. The mines are located particularly in the area along the route of the Chicago, Burlington & Quincy Railroad from Gillette, Wyo., to a point approximately 15 miles northeast of Sheridan, Wyo. Fig. 2 is a graph of coal production in the state of Wyoming from 1918 to 1951.

Across the Big Horn Mountains lies the Big Horn Basin, situated between the Big Horn Mountains and the Rocky Mountains proper. Mining in the Big Horn Basin is almost entirely confined to a small area known as the Gebo Field, near the town of Kirby, which is located on the Chicago, Burlington & Quincy Railroad.

This paper will describe present operating areas, making only brief reference to the history of mining and marketing primarily because of the large area to be discussed.

The coal beds presently mined in the Powder River Basin are all of Tertiary age and are of Fort Union and Wasatch formations of Paleocene and Eocene ages, see Fig. 3. The Fort Union formation, consisting of 2000 to 3200 ft of alternate sandstone, shale, and coal, is divided from oldest to youngest into the Tullock, Lebo Shale, and Tongue River members. Most of the mineable coal is in the Tongue River member, which is 500 to 800 ft thick.

The Carney coal bed, varying from 7 to 20 ft in

thickness, is considered the base of the Tongue River member. About 85 ft above the Carney is the Monarch bed, ranging from 18 to 42 ft in thickness. At approximately 210 ft above the Monarch is what is locally known in the Sheridan district as Dietz No. 3 bed, which varies from 12 to 30 ft in thickness. One hundred feet above the Dietz No. 3 is Dietz No. 2 bed, which is 7 to 12 ft thick. At about 100 ft above the Dietz No. 2 is Dietz No. 1 bed, 7 to 15 feet in thickness. Two hundred and fifteen feet above Dietz No. 1 is the Smith bed, approximately 5 ft thick. One hundred twenty-five feet above the Smith is the Roland bed, which is estimated to be approximately 13 ft thick. The Roland bed is considered the base of the Wasatch formation, which in the Powder River Basin is 1000 to 3500 ft thick and is composed largely of shale, sandstone, and coal.

The foregoing measures are the only ones above the Tongue River member that have been mined in the Sheridan field. To the east of the Sheridan field, particularly in the area of Gillette, the Roland bed is considerably thicker and is considered the thickest and most extensive coal bed in Wyoming, reaching a maximum thickness of 106 ft.

From 125 to 225 ft above the Roland lies the Arvada bed, average thickness 9 ft. Three hundred seventy-five to four hundred feet above the Arvada is the Felix bed, which reaches its maximum thickness of 30 ft near Echeta on the Chicago, Burlington & Quincy Railroad. East of the Powder River the bed averages more than 10 ft in thickness, but it thins in a northwesterly direction.

The highest mineable coal in the Powder River field is the Healy bed, about 400 ft above the Felix bed and exposed only in the highest parts of the area where much of it has been burned along the outcrop. Where it has not been burned, the thickness at most of the outcrops is 10 to 15 ft.

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Discussion on this paper, TP 3615F, may be sent (2 copies) to AIME before Feb. 28, 1954. Manuscript, Jan. 16, 1953. Los Angeles Meeting, February 1953.

The foregoing comprise the major coal beds above the Tongue River member; however, there are a number of intermittent and upper beds not being mined at the present writing.

Below the Tongue River member and in the Sheridan field lying 60 ft below the Carney bed is Masters No. 1 bed, 6 to 16 ft thick; 60 feet below the Masters No. 1 bed is the Masters No. 2 bed, which is approximately 12 ft thick.

The coal throughout the Powder River region is classed as subbituminous C and B rank, as determined according to the standard specifications of the American Society for Testing Materials, see Table I. A small area in the northeastern part of the Powder River Basin is believed to contain lignite and is very near in rank to subbituminous C coal. In earlier years, higher quality coal in the Black Hills region, known as the Cambria field, was mined and coked to some extent. This field, however, is foreign to the Powder River Basin in that it is of lower Cretaceous age. Most of the coal in the Cambria field is of high volatile C bituminous rank. Coal from certain parts of the bed has coking qualities and has been used in the past for manufacturing coke for smelters in the Black Hills region. The mines in the Cambria field have been closed for a number of years, and most of the better quality and more ac-

ceptable coal has been mined out. There is no other known coking coal in northern Wyoming. There is possibility of coking coal lying at great depths in beds that have not to date been explored.

Mining Operations, Powder River Basin, Wyoming

Present mining operations in the Powder River Basin are confined to the Wyodak Coal Co., operating about 6 miles east of Gillette; the Sheridan-Wyoming Coal Co., Inc., operating 12 miles northwest of Sheridan; and the Big Horn Coal Co., operating approximately 14 miles northwest of Sheridan. These companies are the only rail shippers now in the Powder River Basin. They are served by the Chicago, Burlington & Quincy Railroad.

In addition to the foregoing are the following truck mines in Sheridan County: the Custer Coal Co., the Storm King Coal Co., the Black Diamond Coal Co., and the Welch Coal Co. There are numerous so-called *country-bank* mines in the Powder River Basin operating on a very small scale.

There are numerous locations of readily available coal from the various beds throughout the entire area. It is only the lack of available market that prohibits exploitation on a larger scale. In the early days of mining in the Powder River region the coal was used primarily for railroad fuel and domestic



Fig. 1—Location of the larger producing coal areas of northern Wyoming. Heavy shading indicates subbituminous coal areas containing beds more than 30 in. thick. Light and intermediate shading indicates subbituminous areas containing beds less than 30 in. thick or of unknown thickness. Intermediate shading also indicates coal-bearing formations less than 30 in. thick.

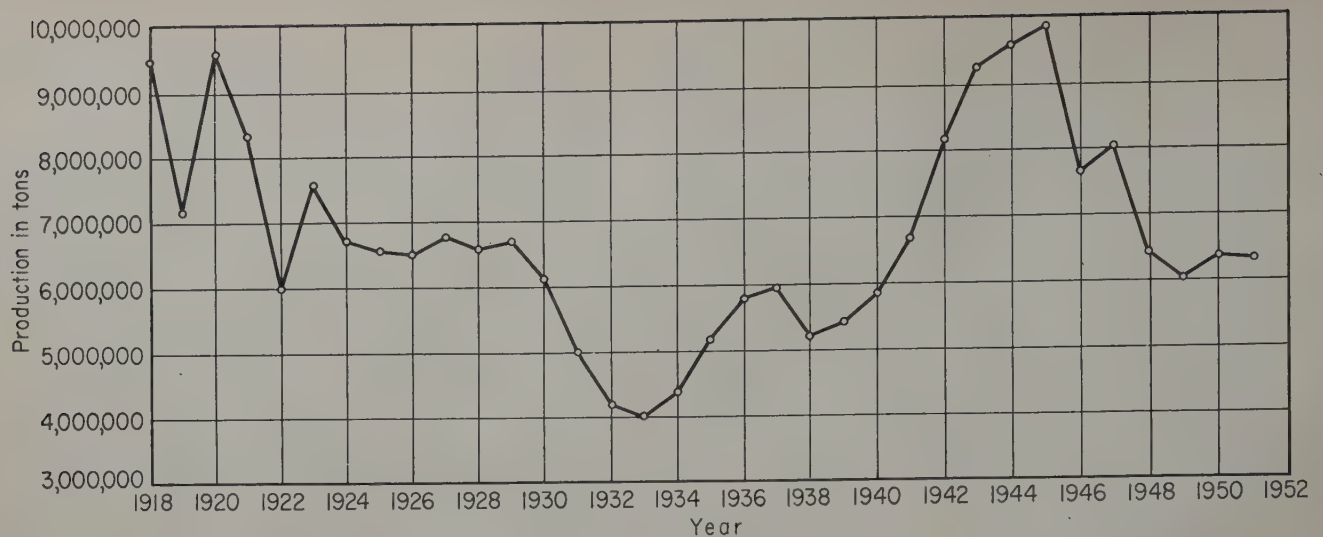


Fig. 2—Coal production in the state of Wyoming from 1918 to 1951. Source of information is the annual report of the State Inspector of Coal Mines of Wyoming, 1951.¹

consumption by local ranch and farm owners and was sent to the very sparsely settled communities.

Table I. Data from *Classification of Coals by Rank*. Determined According to the Standard Specifications of the American Society for Testing Materials as Applicable to Subbituminous and Lignite Coals.

Class	Group	Limits of Fixed Carbon or Btu Mineral-Matter-Free Basis	Requisite Physical Properties
Subbituminous	Subbituminous A coal	Moist Btu, 11,000 or more and less than 13,000*	Both weathering and nonagglomerating
	Subbituminous B coal	Moist Btu, 9500 or more and less than 11,000*	
	Subbituminous C coal	Moist Btu, 8300 or more and less than 9500*	
Lignite	Lignite	Moist Btu, less than 8300	Consolidated
	Brown coal	Moist Btu, less than 8300	Unconsolidated

* Coals having 69 pct or more fixed carbon on the dry, mineral-matter-free basis shall be classified according to fixed carbon, regardless of Btu.

The most easterly operation is the Wyodak strip mine, believed to be the Roland and Smith beds which have merged to form an aggregate thickness of about 90 ft with only an 8-in. parting separating the two beds. The analysis of this coal, according to the U. S. Bureau of Mines, is as follows:

Moisture, pct	31.8
Volatile matter, pct	30.1
Fixed carbon, pct	31.6
Ash, pct	6.5
Btu as rec'd	7,840
Sulphur, pct	0.6

The coal is marketed industrially to cement plants, power plants, and to the Chicago, Burlington & Quincy Railroad for engine fuel, and domestically in the vicinity of Gillette, Wyo., and in the Black Hills area of South Dakota.

The operation consists of stripping an overburden of from 10 to 30 ft, exposing the 90-ft coal seam, which is strip-mined in conventional manner by means of shovels loading into trucks, discharging into a chunk sizer in a central head house from where the coal is conveyed by rubber belt to the

preparation plant. Because of the cleanliness of the coal seam, washing or picking is not required; thus the preparation plant entails normal sizing, dedusting, and oiling. Tonnage from Wyodak mine from 1930 to 1951 inclusive is shown in Table II.

Coal in the Sheridan field is mined both by strip mining and underground mining. The Big Horn Coal Co., mining variably in the Monarch and Carney beds and in what is believed to be the Upper Masters bed, applies strip-mining methods only. A general picture of its operations entails removing overburdens from 10 to 85 ft with large Diesel drag-line equipment, having boom lengths up to 130 ft and buckets up to 6 cu yd in size. Some of the overburden requires blasting, but most of it does not and is easily removed. The coal is cut along the high wall with a vertical saw to improve size consist. The coal is blasted and then loaded out with a 4-cu yd power shovel into 15-ton trucks and hauled an average distance of approximately 5 miles to a preparation plant on the main line of the railroad.

The preparation plant consists of a chunk sizer and main screens and rescreens to make standard domestic preparation, as well as industrial and railroad fuel preparation. No picking or washing is required because of the cleanliness of the seam and selective mining. The Carney seam contains a clay parting located approximately in the center of the seam and varying in thickness from a lens to 8 to 10 in.; when this is encountered the coal is taken in two lifts, the top half being removed and the parting cleaned off prior to removing the lower section of the seam.

The Big Horn Coal Co. began operations for the first time in 1943. The Sheridan-Wyoming Coal Co., Inc. came into being in 1920 as the result of a con-

Table II. Wyodak Tonnage, 1930 to 1951

Year	Tonnage	Year	Tonnage
1930	86,611	1941	122,426
1931	92,257	1942	121,435
1932	98,051	1943	119,468
1933	92,136	1944	105,258
1934	113,600	1945	98,355
1935	120,502	1946	174,337
1936	112,782	1947	209,273
1937	112,142	1948	263,266
1938	105,677	1949	314,196
1939	107,923	1950	349,559
1940	118,562	1951	307,074

solidation of six companies operating variously in the Dietz beds and the Monarch and Carney beds. Diminishing market shortly after 1920 forced the closing of several mines, finally confining activities to mining in the Monarch only. At the present time all coal is being produced from one underground mine in the Monarch bed, with a strip operation in the Carney bed as standby for additional tonnage as required. The method of mining entails mining 12 to 16 ft of the Monarch seam on 50-ft centers with room depths to 400 ft. The top coal must be left for roof protection. The top portion of the Monarch

Table III. Analyses of Coal Beds Mined in the Sheridan Field

Bed	Moisture, Pet	Volatile Matter, Pet	Fixed Carbon, Pet	Ash, Pet	Sulphur	BTU
Dietz No. 1	24.7	37.6	33.00	4.7	0.4	8900
Dietz No. 2	22.5	35.3	34.50	6.6	0.9	9040
Dietz No. 3	24.1	31.3	38.5	6.1	0.6	8820
Monarch	22.56	35.49	38.4	3.55	0.52	9502
Carney	26.2	30.2	39.2	4.4		9454
Masters Upper	23.9	34.1	38.2	3.8	0.3	9300

bed is interspersed with impurity bands varying from carbonaceous shale to soft clay. The overlying strata is variable sandy shale, shale, and clay beds which are very unstable. The top portion of the Monarch is of inferior quality and as such cannot be marketed. Pillars are recovered on the retreat, and extraction of the mined portion of the bed is approximately 85 pct. The coal is cut with shortwall machines and is drilled with mobile rubber-tired drill equipment. *Airdox* is employed to dislodge the coal, which in turn is loaded out with mobile loading equipment into 10 and 12-ton shuttle cars discharging directly into 3 and 5-ton pit cars, which are hauled to the preparation plant, located on the railroad approximately three-quarters of a mile from the mine portal, by 12 and 15-ton locomotives.

The preparation plant entails main screens, re-screens, chunk sizers, slack sizers, and oil treating plant. Here again, no picking or washing is required because of the cleanness of the coal mined.

Analyses of coal beds mined in the Sheridan field are given in Table III.

The market from the mines in the Sheridan field is common to all producers: domestic consumption,

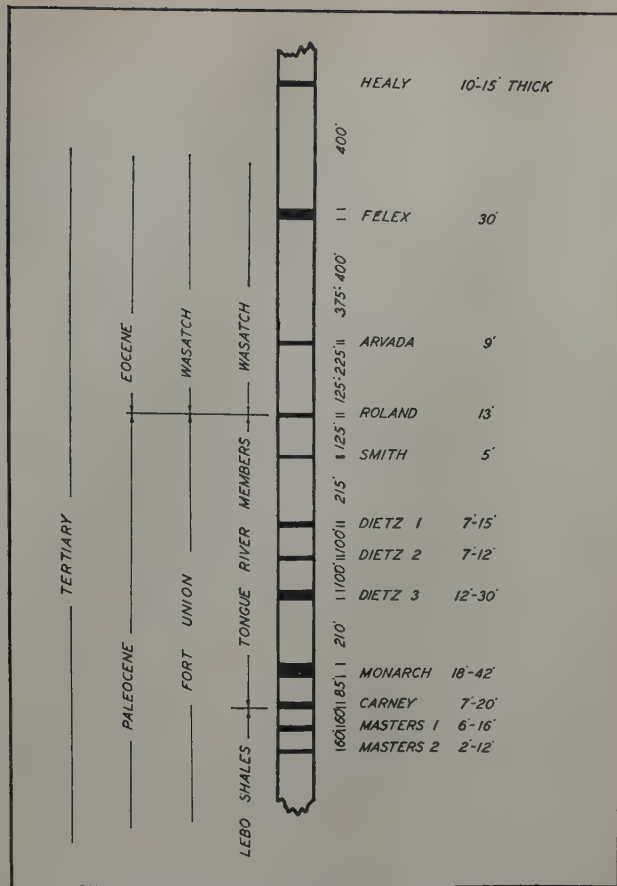


Fig. 3—A stratigraphic chart showing the relation of coal members in the Powder River Basin, Wyoming.

power plants, sugar factories, State and Federal institutions, including Indian agencies, and engine fuel for the Chicago, Burlington & Quincy Railroad.

Tonnages from the various mines in Sheridan County from 1930 to 1951, inclusive, are shown in Table IV.

Many inroads have been made into the coal markets by competitive fuels. The development of oil and gas fields in Wyoming and Montana particularly have brought about considerable conversion to oil and gas in industrial and domestic outlets. In addition, the dieselization program of the Burlington

Table IV. Coal Production in Sheridan County, Wyo., 1930 to 1951. From Annual Report of the State Coal-Mine Inspector

Year	Sheridan-Wyoming Coal Co., Inc.	Big Horn Coal Co.	Custer Coal Co. (Armstrong)	Storm King Coal Co. (Black Diamond)	Welch Mine	Star Mine	Total, Sheridan County, Including Wagon Mines
1930	702,226			8,670		8,369	723,338
1931	539,921		3,955	9,250		9,295	565,809
1932	454,083		8,784	11,500		10,695	486,696
1933	412,652		6,716	9,587		11,253	441,405
1934	439,962		2,525	9,711		9,198	466,104
1935	519,157		2,334	8,703		9,253	543,032
1936	642,079		3,510	10,684		7,682	668,514
1937	625,219		1,990	14,423		6,345	651,664
1938	603,599		2,049	14,870		5,721	629,792
1939	557,124		1,723	13,640		7,037	584,183
1940	572,978		1,999	16,512		6,917	606,583
1941	570,025		2,502	15,838		7,175	602,069
1942	754,469		2,308	16,384		8,487	786,251
1943	935,641		2,645	23,514			962,681
1944	1,010,737	107,201		21,906			1,140,658
1945	1,004,540	264,723		20,696			1,290,672
1946	828,268	483,576	1,375	17,074			1,330,877
1947	612,064	551,429	702	16,716			1,181,252
1948	564,551	464,331	2,236	17,280			1,048,713
1949	690,910	425,663	1,920	17,050	12,372		1,147,916
1950	494,622	350,841	4,045	13,674	15,684		878,866
1951	326,826	321,011	2,035	13,219	15,324		680,415

Railroad has reflected a definite reduction in railroad fuel coal consumption.

Big Horn Basin, Northwestern Wyoming

The Big Horn Basin, which includes about 4400 square miles in Big Horn, Park, Hot Springs, and Washakie Counties, is a broad structural basin bounded by the Big Horn Mountains on the east, the Owl Creek Mountains on the south, and the Absaroka Range on the west. The Big Horn Basin is a region of broad dissected plains characterized locally by badland topography. Around the margin of the basin, particularly on the southwest side, the rocks are folded locally into anticlines and synclines. Where coal-bearing rocks are exposed in the anticlines, erosion of the central parts of the structures has resulted in irregular patterns of coal distribution. The coal-bearing rocks in the Big Horn Basin are of Upper Cretaceous age and comprise the Mesaverde, Meeteetse, and Lance formations, and also Fort Union formation of Paleocene age.

The present operations in the Big Horn Basin are in the Gebo field located in Hot Springs County at the southern end of the Big Horn Basin west of the Big Horn River. The coal occurs in the Mesaverde formation of Upper Cretaceous age in a zone between two masses of sandstone. The most important bed is the Gebo, which has a maximum thickness of 11 ft and is more than 5 ft thick along the outcrop. Several other beds, less extensive, are of mineable thickness. The maximum dip is about 25°.

The coal in the Gebo field is of subbituminous B and A rank. An average analysis on the *as-received* basis from the Gebo bed shows:

Moisture, pct	9.9
Volatile matter, pct	36.2
Fixed carbon, pct	39.8
Ash, pct	4.1
Btu	11,760

Mining is generally pursued by either hand loading or shaking conveyors, loading the coal out of rooms which are driven up the dip. The coal is discharged into pit cars on entries driven on the strike. The coal is hauled to partings just off the main slopes, which are generally driven directly on the dip. Rope haulage is used to haul the coal to the tippie. The preparation plants are common main screen affairs, preparing the coal into the simple domestic sizes, such as Lump, Egg, Nut, and Stoker coal. In one instance the coal is prepared at the mine, hauled by truck approximately 5 miles, and loaded into railroad cars. In the other instance, the mine run coal is hauled from the mine to a tippie on the railroad, where it is prepared and discharged directly into the railroad cars.

Lack of market prevents proper exploitation of this field. In 1951 the following companies reported tonnages as follows:

Mine	Tons
Haverlock and MacCullum, Valley Coal Mine	4011.52
Roncco Coal Co., Roncco Coal Mine	8765.00
Sheridan-Wyoming Coal Co., Inc. Miller Mine	9510.90
Vlastos Emanuel, Osborne Mine	2179.30
Total	24,466.72

This tonnage indicates a decrease of 12,532.23 tons less than tonnage for 1950.

The market history of the Gebo field is parallel to that of the Powder River Basin, namely, sugar fac-

tories, railroad fuel, small power plants, and local domestic outlet.

Mining conditions are generally more difficult in the Big Horn Basin because of folding, faulting, and general inconsistencies of the coal seam. In recent years gas and oil discoveries and subsequent production have cut serious inroads into the coal market from the Big Horn Basin. Of the companies reporting tonnages for 1951, the Roncco Coal Co. and the Sheridan-Wyoming Coal Company, Inc. were the only rail shippers. In 1952 the Sheridan-Wyoming Coal Co., Inc. sold its interest to the T & T Coal Co., which is now shipping by rail over the facilities formerly owned by the Sheridan-Wyoming Coal Co., Inc. Production tonnages for 1938 to 1951 are given in Table V.

Red Lodge Field, Southwestern Montana

To the north of the Big Horn Basin and in the southwest corner of Montana lies the Red Lodge field, Fig. 4. The first mine in the Red Lodge District was opened previous to 1882, but up to 1889 opera-

Table V. Production Tonnages, Big Horn Basin, 1938 to 1951

Year	Tonnage	Year	Tonnage
1938	64,206	1945	160,630
1939	65,392	1946	72,480
1940	75,998	1947	69,502
1941	91,909	1948	50,794
1942	137,916	1949	48,430
1943	174,997	1950	24,467
1944	155,387	1951	24,467

tions were conducted on a small scale. Operations were increased from 1889 to 1905, in which year nearly 600,000 tons were mined. Tonnage has varied upward and downward considerably from that time to the present. Tonnages for the years 1927 to 1950 are shown in Table VI.

Geology: Referring to the lower lefthand corner of Fig. 4, the Red Lodge field is at the foot of the Bear Tooth Mountains in Carbon County, and the coal occurs in the Fort Union formation of Tertiary (Paleocene) age. The Fort Union formation in the Red Lodge field is composed mainly of sandstone and shale and is believed to be about 5000 ft thick.

Table VI. Production Tonnages, Red Lodge Field, 1927 to 1950

Year	Tonnage	Year	Tonnage
1930	335,081	1940	368,975
1931	261,809	1941	402,324
1932	278,321	1942	622,678
1933	341,388	1943	527,607
1934	273,456	1944	627,617
1935	289,702	1945	555,979
1936	301,932	1946	334,571
1937	361,918	1947	265,693
1938	301,048	1948	235,326
1939	327,996	1949	220,341
		1950	196,626

It contains in its upper one-third a coal-bearing zone 825 ft thick between an upper and lower-bearing member. The coal is considered high volatile C bituminous rank because of its resistance to weathering.

In the vicinity of Red Lodge eight beds of coal have been worked. They are as follows.

Bed No. 1 contains 7 ft of coal overlain by carbonaceous shale and underlain by 4 to 6 in. of shale above sandstone.

Bed No. 1½, 5 ft of bright coal of many thin partings, has a sandstone roof and shale floor.

Bed No. 2 contains 8 ft of coal in six benches with partings of shale $\frac{1}{2}$ to 1 in. thick. It rests upon shale and is covered by sandstone.

Bed No. 3 contains 10 ft of dirty coal.

Bed No. 4 has 10 ft of good coal occurring in three benches with a parting 1-in. thick 23 in. below the top. This bed is underlain by sandstone and covered by gray shale.

Bed No. 4 $\frac{1}{2}$ is composed of 3 $\frac{1}{2}$ ft of coal occurring in several benches and has shale above and below.

Bed No. 5 contains 12 ft 1 in. of coal in four benches, with partings $\frac{1}{2}$ to 2 in. thick, and lies between beds of hard shale.

Bed No. 6 has 4 ft 11 in. of coal in a single bench with sandstone roof and floor.

The coal below Bed No. 6 is placed in one group because the interval between the beds is not sufficient to allow any one coal to be mined without disturbing those above. Most of the coal mined has been obtained from Beds 2, 3, and 4.

The coal beds dip irregularly from the outcrop on the east side of the field to the southwest at the town of Red Lodge. The dip is 18°, decreasing toward the southeast to 9° in the western part of the Bear Creek district and to 3° a short distance farther south. The coal beds are terminated to the west by the Bear Tooth Mountain fault whose eastern side is down-thrown 500 to 600 ft.

A typical analysis, on an *as-received* basis, of coal from Bed No. 4 at Red Lodge shows a heat value of 10,330 Btu., 12.7 pct ash, and 1.3 pct sulphur.

Mining: The two major operators in the Red Lodge field today are the Montana Coal & Iron Co., Foster Mine, and the Brophy mine of the Brophy Coal Co. The Foster mine is situated 1 $\frac{1}{2}$ miles south of Bear Creek, Carbon County, Mont., and is served by the Montana, Wyoming & Southern Railroad. This mine is opened by three slopes in the No. 3 coal bed, which averages 78 in. in thickness and dips about 5° in a southwesterly direction. This mine employs approximately 85 men with an average daily production of nearly 600 tons of coal, all of which is top cut and sheared at the working faces with a track-mounted Universal mining machine and loaded directly into 5-ton drop bottom cars with track-mounted and crawler-type loading machines. The estimated life of the mine is approximately 50 years at the present annual rate of production. The total production for 1951 was 134,664 tons. The production from Jan. 1 to July 31, 1952, was 62,712 tons.

The mine is developed by room and pillar method but pillars are not recovered. Entries are driven from 12 to 14 ft in width on 50 and 80-ft centers. The immediate roof throughout the mine consists of friable shale, ranging from 6 to 18 in. in thickness, and the main roof consists of hard shale in some areas and a hard firm sandstone in others. Legs and crossbars are generally used to support the roof in the rooms along the haulageways and in the air courses. Where crossbars are not required, props and cap pieces are used. Trolley locomotives are used to transport trips along the main haulageways, including the haulage slope, and cable reel locomotives are used to service the loading machine in each working section. The coal is hauled to a tippie consisting of a crossover dump cascading coal on to shaking and vibrating screens. Crushing equipment, a dry cleaning plant, and wet washing plant are included to permit all marketable sizes of coal to be prepared.

An average analysis of the coal is as follows:

Moisture, pct	11.1
Volatile matter, pct	34.5
Fixed carbon, pct	39.2
Ash, pct	6.2
Btu	11,190
Sulphur, pct	1.4

The Brophy mine is located about 6 miles east of Red Lodge, Carbon County, Mont., and is served by the Montana, Wyoming & Southern Railroad. The mine was opened in 1922 by the Smokeless and Sootless Coal Co. The present operating company acquired the mine in 1940. The mine is opened by two slopes in the No. 2 coal bed, which averages 74 in. in thickness and dips about 3 $\frac{1}{2}$ ° in a southerly direction. Approximately 25 men are employed, and the average daily production is approximately 200 tons of coal.

The coal is undercut at the working faces with shortwall mining machines. Coal is broken down with *Cardox* and loaded directly into shuttle cars with a crawler-type loading machine. Total production in 1951 was 15,249 tons.

The mine is developed by room and pillar method; pillars are recovered by open-end methods and by slab cutting the inby ends with a shortwall mining machine.

The immediate roof overlying the coal bed consists of hard dark shale ranging from 4 to 6 ft in thickness, and the main roof is about 60 ft of hard sandstone. The immediate roof is very friable and generally requires legs and crossbars for support along the haulageways in the rooms and the air courses. Where crossbars are not needed, props and cap pieces are used.

A trolley locomotive is used to haul trips from the loading points to a parting at the bottom of the slope from where an electric hoist, located on surface, hauls the coal to the tippie where the coal is dumped through a crossover dump on to the shaker screen. Tippie equipment consists of shaker and vibrating screens, washing plant for Pea coal, and crusher for sizing.

A representative analysis of the field on the *as-received* basis is as follows:

Moisture, pct	9.5
Volatile matter, pct	35.9
Fixed carbon, pct	48.3
Ash, pct	6.3
Btu	10,603

Market outlets from the two foregoing mines are to the Montana, Wyoming & Southern and the Northern Pacific Railroads, State Institutions, small industrial plants, and the general domestic trade in Washington, Idaho, Montana, and North Dakota.

There are several smaller mines, considered as wagon mines, operating in the Bear Creek field and serving some of the local market. These mines are very low in production, and are therefore not mentioned here.

Bull Mountain Field, Central Montana

Geology: Referring to Fig. 4, the Bull Mountain field, located mainly in Musselshell and Yellowstone Counties, is at present the largest commercial producing area in Montana. The coal-bearing rocks in the field are the Tongue River member of the Fort Union formation and to a lesser extent the underlying Lebo shale member of the Tullock formation, see Fig. 5.

The coal is of subbituminous B and subbituminous A rank. Most of the mining is in the Roundup bed, which crops out in the north and northwest part of



Fig. 4—Location of major producing coal areas in the state of Montana. Heavy shading in center, upper right, and lower right indicates subbituminous areas containing beds more than 30 in. thick. Heavy shading in upper left indicates bituminous coal beds more than 14 in. thick. Intermediate shading in the Bull Mountain field, in the Garfield County field, and in the narrow strip extending south from the Garfield County field indicates subbituminous coal beds less than 30 in. thick. Intermediate shading to the extreme right indicates beds of lignite. Intermediate shading at upper left indicates bituminous coal less than 14 in. thick.

the field. In the vicinity of Roundup this bed is 500 ft above the base of the Tongue River member and ranges in thickness from $3\frac{1}{2}$ to 6 ft. This thickness decreases to the east gradually at first and then sharply until at a point about 12 miles from Roundup it measures only 1.6 ft. A parting occurs in the northwestern part of the field and is variable in thickness from a trace to approximately 24 in. There are from 2 to 3 ft of coal above the parting and approximately 3 ft below the parting.

The Carpenter Creek bed is about 50 ft below the Roundup bed and is commercially important in the northeastern part of the field, where it ranges in thickness from 4 to 8 ft. The Lebo shale member of the Fort Union formation contains, for the most part, only carbonaceous shale in the Bull Mountain field.

The Big Dirty coal bed, which occurs near the middle of the member, ranges in thickness from 22 to 24 ft, but although it is a prominent marker in the field, it contains only 2 in. to 1 ft of clean coal. Economically this bed is unimportant.

The Mammoth and Rehder beds are two coal beds separated by an interval in places so slight that the coal can be mined as a single bed; in other places the interval is as much as 25 ft. The Mammoth outcrops in Township 7 North, Ranges 26, 27, and 28 East, and in Township 6 North, Ranges 26, 27, and 28 East.

Mining: The larger producers in the field are

Sheridan-Wyoming Coal Co., Inc., Mine No. 3, located one mile west of Roundup; the Republic Coal Co., Klein mine No. 2, located at Klein, Musselshell County, operating in the Roundup bed; Bair-Collins Coal Co., located about 5 miles southeast of Keene, Musselshell County, operating in the Carpenter Creek bed; and the Jeffries Coal Co., Carlson-Jeffries mine, located 7 miles southeast of Roundup, Musselshell County, operating in the Mammoth bed.

The foregoing mines comprise the rail shippers in the Bull Mountain field. There are numerous small so-called wagon mines operating in the Roundup and Mammoth beds which will not be mentioned here because of their small tonnages.

The No. 3 mine of the Sheridan-Wyoming Coal Co., Inc. is opened by two vertical shafts and four slopes in the Roundup coal bed, which ranges from 46 to 66 in. thickness, and dips about $\frac{1}{2}^\circ$ in the present workings in a southeasterly direction. The main haulage slope, which follows the coal from the crop, has a considerable catenary, ranging from 26° at surface to $2\frac{1}{2}^\circ$ at the bottom of the slope, which is approximately 3000 ft in length; thence in by to the present workings the grade lessens to $\frac{1}{2}^\circ$.

About 140 men are employed, producing an average of 950 tons per day. The coal is undercut with shortwall mining machines and drilled with hand-held drills. The coal is dislodged with *Airdox*, and

the following combinations of equipment are used to load the coal into pit cars: 1—Crawler type loading machines loading directly into pit cars. 2—Crawler type loading machines discharging into chain conveyors discharging on to rubber belts discharging into pit cars. 3—Crawler type machines discharging to shuttle cars discharging to pit cars.

The 2½-ton pit cars are hauled in 50-car trips by a 36-ton locomotive to the slope bottom parting. An electric hoist, located on surface, hauls the cars up the slope to the tippie in 11-car trips. A rotary dump empties the cars into the tippie from where the coal can be diverted to a chunk sizer, or discharged directly over the main shaker screens where the +2 in. is sized out and passed over picking tables, served by hand pickers, and discharged directly into railroad cars. The -2 in. passes over a secondary rapid shaker screen and is prepared into Nut or Stoker sizes. All sizes can be oil treated.

The total production for 1951 was 149,092 tons; the production from Jan. 1 to Aug. 31, 1952, was 67,336 tons. The estimated life of the mine is approximately 50 years at present rate of production.

The mine is developed by room and pillar method but pillars are not recovered. Main entries are driven three abreast and cross entries in pairs. The main roof consists generally of hard sandstone, but in some parts a hard to friable shale ranging from 2 to 24 in. thickness lies between the main roof and the top coal. A bony streak varying from a trace to 3 in. is generally at the top of the coal, requiring picking the coal in the preparation plant. This mine was opened in 1908 and has operated continually since.

The present market outlet consists chiefly of mill and smelter, State and Federal institutions and plants, and domestic outlet in the states of Washington, Idaho, Montana, North and South Dakota, and western Minnesota.

A representative analysis on an *as-received* basis is as follows:

Moisture, pct	11.0
Volatile matter, pct	24.48
Fixed carbon, pct	54.88
Ash, pct	9.64
Btu	11,126

The Klein No. 2 mine of the Republic Coal Co. is the only other rail shipper mining coal out of the Roundup bed at the present time. This company is owned by the Chicago, Milwaukee, St. Paul & Pacific Railroad and is generally considered a captive operation. It is situated at Klein, Musselshell County, Mont., and is served by the above railroad.

The mine is opened by three vertical shafts and by a 45° slope and it is developed in the Roundup coal bed which ranges from 40 to 64 in. thickness and dips about 1° in a southeasterly direction.

Approximately 110 men are employed at the present time with an average daily production of 1100 tons of coal. The total production for 1951 was 405,144 tons.

The mine is developed by room and pillar method but pillars are not recovered. Main entries and cross entries are driven three abreast about 14 ft wide on 50-ft centers. Rooms are driven about 35 ft in width on 46-ft centers off both sides of the cross-entries and are up to 300 ft long.

The immediate roof consists generally of very soft shale ranging from 22 to 24 in. thickness. In some sections this shale disintegrates rapidly, but in other places it is firm and requires only the regular

number of timbers for support. The main roof is generally hard, firm shale, and in some sections massive hard sandstone. The coal is undercut with shortwall mining machines and is drilled with both hand-held and post-mounted drills. Coal is blasted on shift with permissible explosives and loaded out with mobile-type loading equipment. Shaker conveyors had been used in one section of this mine but are not being used at the present time. The coal is loaded directly into 2-ton pit cars and hauled to the shaft bottom by trolley locomotive. An electric hoist on surface provides motive power for hoisting the pit cars to the automatic dump located at the head-house.

Preparation is achieved in a conventional-type tippie, composed of shaker screen, vibrating screen, crushers, mixing conveyor, and picking table. No effort is made to prepare domestic coal at this mine, practically all of the production being prepared for railroad engine use and industrial use.

A representative analysis on an *as-received* basis is as follows:

Moisture, pct	14.0
Volatile matter, pct	30.8
Fixed carbon, pct	49.5
Ash, pct	5.7
Btu	11,160
Sulphur, pct	0.7

The Carlson-Jeffries mine of the Jeffries Coal Co. is the only other rail shipper in the immediate

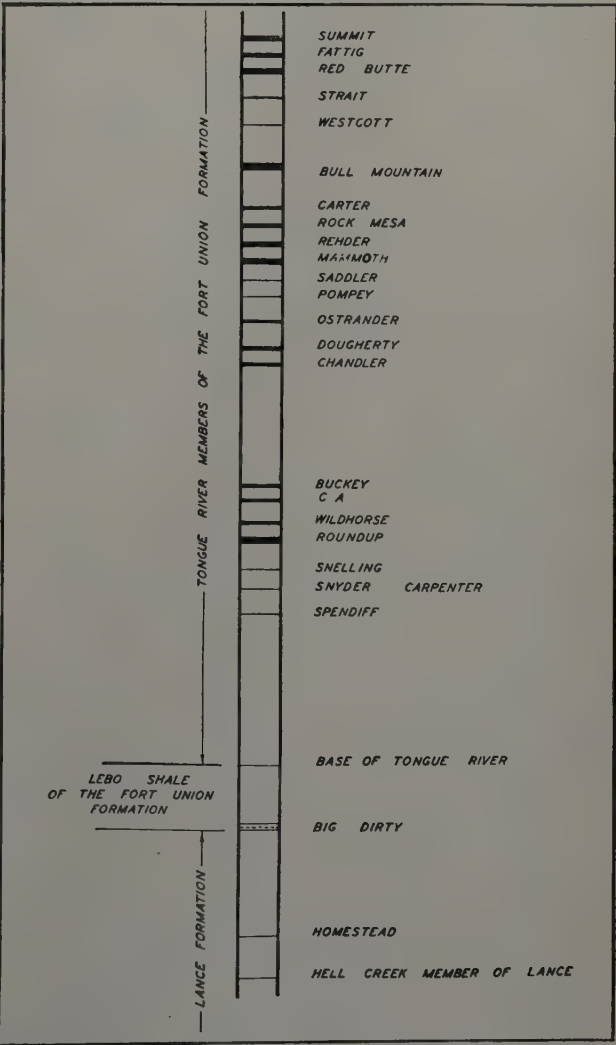


Fig. 5—Coal members in Bull Mountain Field, Montana.

vicinity of Roundup. It is served by the Chicago, Milwaukee, St. Paul & Pacific Railroad.

The mine is opened by three short slopes and two vertical shafts in the Mammoth coal bed, which averages 9 ft in thickness and lies practically level in the area being mined. When this mine was last operated it employed 19 men and produced approximately 150 tons of coal per day, all of which was undercut and sheared with a track-mounted Arc-Wall mining machine and loaded with a crawler-type loading machine. This mine produced in the neighborhood of 20,000 tons per year.

The mine was developed by room-and-pillar method but pillars were not recovered. Entries were driven two abreast about 18 ft in width and 45-ft centers. Rooms were driven about 26 ft in width on 35 to 38-ft centers. The immediate roof consists of unmarketable coal ranging from 3 to 4-ft in thickness. The main roof consists of hard shale about 25 ft in thickness, which is overlain by massive sandstone. The coal was blasted with permissible explosives.

The coal was loaded into pit cars and hauled by a combination storage battery and trolley locomotive from the underground working sections up the slope to the surface to a storage bin. The coal was trucked from the storage bin to the tippie on the railroad, approximately 6 miles distant, where it was prepared into standard domestic sizes for market outlet similar to that described for the Roundup No. 3 mine. At present writing, lack of market has caused this mine to be idle this year.

The Keene No. 2 mine of Bair-Collins Coal Co., located about 5 miles southeast of Keene, Musselshell County, is served by the Chicago, Milwaukee, St. Paul & Pacific Railroad. It is opened by a single entry driven in a southeasterly direction and a vertical shaft 10 ft square and 60 ft deep, and by a single slope driven in a southern direction a distance of 100 ft on the dip of the coal bed, which is the Carpenter Creek coal bed. This bed ranges from

96 to 102 in. thickness; it dips about 3° in a southerly direction.

Forty-five men are employed at the present time, and daily production averages 500 tons of coal. The total production for 1951 was 59,330 tons.

The mine is developed by a room and pillar method but pillars are not recovered. Main and cross entries are driven four abreast and panel entries are driven in pairs 16 ft in width on 65-ft centers. Rooms are driven 26 ft wide on 55-ft centers to a maximum depth of 300 ft. The roof overlying the coal bed consists of hard sandstone about 40-ft thick. Coal is undercut and sheared with a combination top cutting and shearing machine, track-mounted. The coal is dislodged with *Cardox* and is loaded out with a mobile-type loading machine directly into pit cars. Combination trolley and cable reel locomotives are used to serve loading machines at working faces, and a trolley locomotive is used to transport the mine cars along the main and secondary haulage. The coal is discharged at the surface into a bin from where it is hauled in 20-ton trucks to the tippie, 5 miles distant, on the railroad, where it is dumped into a large bin. The coal is conveyed by a 30-in. conveyor belt from the dump to the tippie.

The tippie consists of one triple-deck shaking conveyor, preparing all marketable domestic and industrial sizes.

The market outlet consists of the same types and areas described for No. 3 mine of the Sheridan-Wyoming Coal Co., Inc.

Powder River Basin in Montana, Forsyth Field, Southeastern Montana

Geology: The only other producer in Montana of any consequence at the present time is the Rosebud open pit of Foley Brothers, Inc., located at Colstrip, Rosebud County, Mont. This mine is considered a captive mine in that it supplies coal only for the Northern Pacific Railroad Co. It is an open-pit

Table VII. Coal Production Record for Red Lodge Field and Bull Mountain Field, Montana, 1930 through 1950. Compiled by Montana Coal Operators Association, Billings, Mont.

Company	1930	1931	1932	1933	1934	1935	1936
Bair-Collins Co.	79,852	87,257	113,100	86,277	85,307	128,166	158,273
Montana Coal & Iron Co.	188,370	155,162	175,259	235,397	225,348	227,705	301,932
Roundup Coal Mining Co.	345,816	122,348	131,243	109,964	136,834	171,955	216,421
Now Sheridan-Wyoming Coal Company, Inc.							
Jeffries Coal Co.					56,584	71,135	72,588
Brophy Coal Co.							
Northwestern Improvement Co.	207,896	105,527	53,534				1,225,467
Republic Coal Co.	410,767	414,416	353,264	352,157	340,141	395,551	442,687
Total	1,232,701	884,710	826,400	783,795	844,214	994,512	2,417,368
Company	1937	1938	1939	1940	1941	1942	1943
Bair-Collins Co.	144,723	124,306	116,596	128,569	128,641	170,621	190,986
Montana Coal & Iron Co.	361,918	301,048	327,996	359,141	387,133	595,986	475,925
Roundup Coal Mining Co.	193,891	124,810	136,913	151,734	148,307	201,772	324,881
Now Sheridan-Wyoming Coal Company, Inc.							
Jeffries Coal Co.	61,994	54,111	44,622	28,647	24,298	51,359	63,200
Brophy Coal Co.				9,834	15,191	26,692	51,682
Northwestern Improvement Co.	1,247,337	1,099,106	1,070,734	1,124,932	1,395,943	1,706,749	2,576,627
Republic Coal Co.	364,259	382,234	411,870	440,314	495,917	502,154	516,806
Total	2,374,122	2,085,615	2,108,731	2,243,171	2,595,630	3,255,333	4,200,107
Company	1944	1945	1946	1947	1948	1949	1950
Bair-Collins Co.	150,695	127,712	62,144	79,626	76,838	94,733	75,000
Montana Coal & Iron Co.	576,003	499,810	298,119	241,318	208,557	196,010	182,654
Roundup Coal Mining Co.	384,397	366,219	226,035	213,388	156,341	112,090	123,891
Now Sheridan-Wyoming Coal Company, Inc.							
Jeffries Coal Co.	46,526	42,860	37,890	47,500	36,130	30,380	20,630
Brophy Coal Co.	51,614	56,169	36,452	24,375	26,769	24,331	13,972
Northwestern Improvement Co.	2,513,385	2,555,208	2,452,753	1,982,185	1,893,051	1,819,343	1,708,149
Republic Coal Co.	531,214	529,815	407,812	470,016	432,559	389,844	270,457
Total	4,253,834	4,177,793	3,521,205	3,058,408	2,830,245	2,666,731	2,394,753

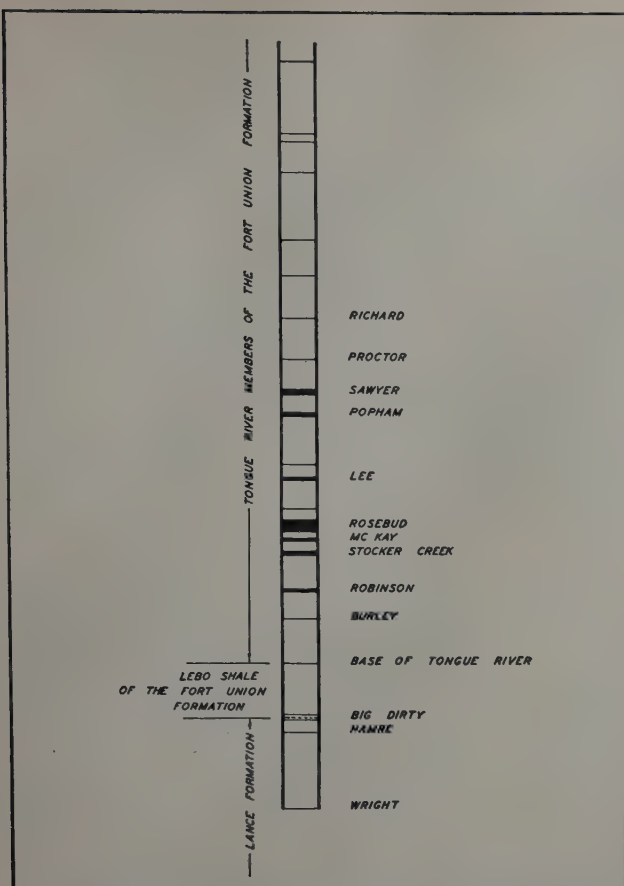


Fig. 6—Chart showing Lance formation and Tongue River members of the Fort Union formation, Forsyth Field, Powder River Basin, Montana.

operation and has been operated since 1924 in the Rosebud coal bed, which averages 26 ft in thickness and dips slightly in an easterly direction.

This bed is 485 ft above the base of the Tongue River member of the Fort Union formation of the Powder River Basin, Fig. 6.

An average of 10,000 tons of coal is produced daily by 88 employees. A total of 1,488,300 tons of coal was produced in 1951, and 746,097 tons were produced during the first seven months of 1952.

Mining: The overburden consists of top shale ranging from 5 to 30 ft in thickness, sand rock from 15 to 30 ft, and shale 30 to 40 ft. The overburden is moved by means of a 13-cu yd and a 20-cu yd electric shovel and relayed with an 8-cu yd dragline. Model D8 bulldozers supplement the stripping shovel operations to remove spillage and other impurities from the top coal bed. The overburden is drilled with churn drills and the vertical holes in the coal are drilled with a jack hammer mounted on a tractor-wagon. All coal is loaded directly into railroad cars by means of a 17-cu yd electric shovel.

Under the present rate of production, this mine is estimated to have a life expectancy of more than 20 years.

Diesel electric locomotives, 600 and 1000 hp respectively, are used to serve the 17-cu yd coal-loading shovels, with trains made up of 12 and 17 railroad cars.

Electric power is received at 69,000 v and transformed to 4100 v for the shovel cables. The shovels are equipped with motor generator sets which convert the 4100 v AC to DC, ranging from 600 to 960 v. All other electrically power-driven equipment is operated at either 220 or 440 v, AC.

An analysis of this coal is as follows:

Moisture, pct	24.4
Volatile matter, pct	28.5
Fixed carbon, pct	39.5
Ash, pct	7.6
Btu	9,080
Sulphur, pct	0.6

In addition to the foregoing described mines, there are approximately 115 small mines scattered throughout the state in the various coal beds and mining tonnages from 10,000 tons annually and less. The total production in Montana, all mines reporting, is estimated at 2,410,994 tons for the calendar year 1951.

The production record shown in Table VII² indicates tonnages reported on the major mines for the period 1930 through 1950.

Vast reserves of readily mineable coal are located in northern Wyoming and Montana. This area is one of the largest coal-bearing areas in the United States. Its potentiality is unlimited. Lack of market outlet is the only reason for the limited exploitation and production of today. The most recent estimate,³ gives the coal reserves in Montana, short tons, as follows:

Bituminous	2,362,610,000	
Subbituminous	132,151,060,000	
Lignite	87,533,270,000	
Total		222,046,940,000 tons

The coal reserves as estimated in Geological Survey Circular 81, Department of Interior, September 1950,⁴ lists the following tonnages in the Powder River Basin and in the Big Horn Basin of Wyoming:

Estimated coal reserves in Montana.		
	Brought forward	222,046,940,000 tons
Bituminous and Subbituminous		
Powder River Basin	94,922,460,000 tons	
Big Horn Basin	581,680,000 tons	
Total		95,504,140,000 tons
Grand total		317,551,080,000 tons

With this grand total of 317,551,080,000 tons in northern Wyoming and Montana, it can readily be seen that this area will prove a valuable asset to the coal industry of America for hundreds of years to come. It is an open invitation to coal-burning industries, steam-generating power plants, chemical plants, and synthetic liquid fuel plants. Impounding of the mountain streams will result in a normal supply of water for vast industries and will aid materially in making use of the great reserve of energy in northern Wyoming and Montana.

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¹ Lyman Fearn: State of Wyoming Coal Mine Inspection Report of 1951.

² Montana Coal Operators Association, Billings, Mont. D. F. Buckingham, Secretary.

³ J. X. Combe, D. M. Brown, H. F. Pulver, and D. A. Taylor: *Coal Resources of Montana*. U. S. Geol. Survey Circ. 53.

⁴ H. L. Berryhill, Jr., D. M. Brown, A. Brown, and D. A. Taylor: *Coal Resources of Wyoming*. U. S. Geol. Survey Circ. 81.

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Localization of Pyrometasomatic Ore Deposits At Johnson Camp, Arizona

by Arthur Baker III

The orebodies are long bedding-plane lenses of chalcopyrite and sphalerite, associated with garnetite masses. Most of the orebodies are within a 50-ft thickness of Cambrian limestone; other Paleozoic limestones and dolomites are locally metamorphosed but only slightly mineralized. Pre-mineral faults are numerous, but shallow folds were the main ore-localizing structures.

JOHNSON camp is in the northwestern part of Cochise County, Ariz., about 50 miles east of Tucson. The nearest major mining districts are Tombstone and Bisbee, respectively 27 and 50 miles to the south, and the Superior-Miami-Globe-Ray porphyry copper group, about 90 miles northwest.

Like many mining camps of the Southwest, Johnson camp is said to have been worked by the Spaniards. The first production on record was in the early 1880's, when an unknown amount of oxidized copper-silver ore in the Peabody mine was mined from replacement orebodies in the Pennsylvanian Naco formation. From 1904 to 1911 outcropping oxidized ores in the Cambrian Abrigo formation were worked, and an estimated 100,000 tons of copper ore were shipped. In 1912 the first large sulphide orebody of the district, the Republic Manto orebody, was discovered, and in the following few years some 250,000 tons of predominantly sulphide ore were shipped. The average grade of this ore was approximately 4.5 pct Cu, 6 pct Zn, 0.8 oz. Ag, and 0.001 oz Au. The Republic, Copper Chief, and Mammoth mines were the principal producers during this period. Mining ceased in the district after 1920, and until 1943 only small-scale leasing operations were carried on.

In 1943 all the mines that had been productive were acquired by the Coronado Copper and Zinc Co., the present operators, and in the 10 years since that time the district production has amounted to approximately 350,000 tons of milling ore averaging 2 pct Cu and 6 pct Zn. Most of this ore was produced from the Republic and Mammoth mines, but since 1950 a large part of the production has been from the new Moore mine.

The total known production from the district, then, is about $\frac{3}{4}$ million tons of copper and copper-zinc ore of low grade. All of this ore was produced from orebodies associated with garnetite in the middle member of the Cambrian Abrigo formation. In addition to this known production, an unknown tonnage of ore was extracted from the Peabody mine orebodies that lie in the Pennsylvanian Naco formation.

Published information on the geology of the district is limited. Aside from brief references in various mining journals, only three papers on the district have been published.¹⁻³ One of these is a U. S. Bureau of Mines report on a diamond drilling program, one is a brief paper on the general geology of the district, and the third is a report on geochemical experiments, with a section on the occurrence of the orebodies. The last two are by John Cooper, of the United States Geological Survey, who has done much detailed work in the area.

Stratigraphy

The rocks of the mineralized area are Paleozoic sediments ranging in age from Cambrian to Pennsylvanian. Several disconformities are present in the stratigraphic column, the most important one being between the Cambrian Abrigo formation and the Devonian Martin formation. There are no angular unconformities. Within the district, the Paleozoic sediments lie in a fairly uniform monocline, striking northwest and dipping 30° to 50° northeast. This local monocline is part of a domal structure centered in the Little Dagoon mountains to the southwest.

The Texas Canyon stock, a quartz monzonite body intruded probably during the Laramide revolution, lies south of the mineralized area (Ref. 2, p. 33). The Paleozoic rocks dip away from the stock, and on the surface are separated from it by at least 1500 ft of Pre-Cambrian rocks. The outcrop pattern of the northeastern edge of the stock suggests that it may dip gently northeastward, passing below the mineralized area at moderate depth. No quartz monzonite has been found in mine workings or diamond drill holes, which reach to depths of 1000 ft. The only igneous rock found in the mineralized area is a lamprophyre dike cutting the Naco limestone in and near workings of the Peabody mine.

With the exception of the lowermost beds—the Bolsa quartzite and the shaly lower member of the Abrigo formation—the Paleozoic sediments are predominantly carbonate rocks, Fig. 1. The middle member of the Abrigo formation, which contains the principal ore-bearing beds, is limestone, with thin shale partings throughout most of its 250-ft thickness. Near the top of this member is a sandy bed some 25 ft thick. The upper member of the Abrigo formation and the lower half of the overlying Devonian Martin formation are dolomitic, with num-

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Discussion on this paper, TP 3684I, may be sent (2 copies) to AIME before Feb. 28, 1954. Manuscript, April 30, 1953. Los Angeles Meeting, February 1953.

erous quartzite and sandy dolomite beds. The upper half of the Martin formation is principally pure dolomite, while the Mississippian Escabrosa formation and the Pennsylvanian Naco formation are chiefly pure limestone.

Small lenses and stringers of ore minerals occur in all the Paleozoic sediments, and in a few places in the Precambrian rocks. Commercial orebodies, however, are found only in the middle member of the Abrigo formation, with the exception of the Peabody mine orebodies in the Naco formation. The scope of this paper is limited to consideration of the orebodies in the Abrigo formation and their associated metamorphic rocks. The metamorphism associated with the mineralization has affected not only the middle member of the Abrigo formation, but also the impure dolomites of the upper member of the Abrigo formation and the lower half of the Martin formation.

Structure

Faulting: The sedimentary rocks are broken and displaced by three groups of fractures: the Northeast faults, the Easter faults, and the Northwest faults. Included in each of these three groups are not only faults with appreciable displacement, but also numerous non-displacing fractures that have the same characteristics as the faults but not the displacement. Included in the Northeast fault group, for instance, are about 30 known faults with displacement, and some 200 identical fractures without displacement.

The Northeast faults are normal faults striking N 15° to 20° E and dipping 65° to 75° E. On the 30 faults of the group that have appreciable displacement, the movement has been directly down the dip of the fault surface. Two faults of the group displace the beds by as much as 200 ft, but for the others the maximum displacement is 40 ft, the great majority having no displacement at all.

Vuggy, symmetrically banded quartz-orthoclase veins up to 1 ft wide are common in Northeast fractures where the walls are either ore or metamorphic rocks, but in unmetamorphosed sediments there is usually only a slightly silicified zone a few inches wide. Chalcopyrite, pyrite, bornite, scheelite, argentiferous tetrahedrite, fluorite, wolframite, and galena have been found in the veins. The last four of these are found only in the Northeast veins; they do not occur in the bedded orebodies. No zinc minerals are found in the veins. The veins are nowhere wide enough to constitute orebodies in themselves, and the sulphides are abundant in them only where the walls of the veins are ore of the bedded orebodies. The sulphides are later than the vein silicates, since they mold around silicate crystal faces and replace silicates along fractures. The walls of most Northeast fractures are chloritized to a depth of about an inch, regardless of whether the fractures carry sulphides or other vein matter. On all the displacing faults, and on many of the non-displacing ones, this chlorite is strongly sheared.

The faults of the Easter group strike N 70° E to S 70° E, and dip either about 45° S or 80° S. Most Easters are normal faults, offsetting the beds by as much as 250 ft, but one major Easter is a reverse fault with offset ranging from 50 to 150 ft. At most places Easter faults consist of a zone of sheared chlorite that is 3 in. to 3 ft thick, with occasional fragments of partially chloritized wall-rock embedded in the chlorite. Where the faults pass through orebodies, this main zone is in places bordered

by a zone in which the ore wall rock has been shattered and then partially replaced by chlorite along the fractures. The chlorite in these zones is sheared like that of the zones of major movement, but the wall-rock fragments have been neither rotated nor moved any appreciable distance. In rare instances, at or near the intersections of the Easter faults with orebodies, chalcopyrite occurs in the fault zones, intergrown with unhealed chlorite. In one small area of an Easter fault zone, molybdenite is found as a replacement of chlorite along the shear surfaces.

Faults of both the Northeast and the Easter groups are distributed uniformly throughout the district. Northwest faults, however, are common only in the western end of the district and are almost entirely absent toward the eastern end. The Northwest faults strike about N 10° W and dip steeply either east or west. The maximum known displacement is 150 ft. The Northwest fault zones locally contain quartz veins similar to those of the Northeast faults, but more commonly they are simply sheared zones a few inches to a few feet in width.

Movement on all three groups of faults was essentially contemporaneous, as is shown by the fact that faults of each group are cut and offset by faults of the other two groups at one place or another in the district. The initial development of the faults began before the metamorphism and mineralization, since faults of both the Northeast and Easter groups locally influenced the distribution of the metamorphic rocks and ore. Ore fragments found in the shear zones of faults of all three groups demonstrate that some of the movement was later than the mineralization. The final fault activity in the district was the movement that produced the 100-ft and greater displacements on the major Easter faults, since these Easters offset all the faults of the other two groups they intersect.

Folding: Superimposed on the regional monocline of the Paleozoic sediments are two sets of shallow

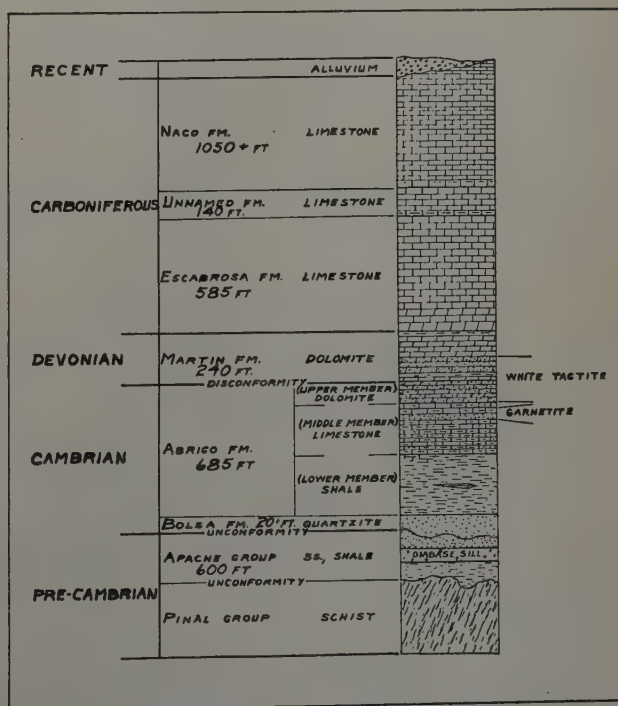


Fig. 1—Diagrammatic section of rocks at Johnson Camp. (After Cooper, Arizona Bur. Mines Bull. No. 156.)

warps, the Manto folds and the Winze folds. The axes of the two sets of folds are at right angles to each other: the Manto folds trend between due East and S 70° E and plunge 15° E, while the Winze folds trend about N 10° W and plunge 30° N. Folds in both groups are best developed in the ore-bearing beds, decreasing in intensity above and below these beds. Although all the folds are very shallow, some of them are at least 700 ft long.

The most completely developed fold in the district is the Republic Manto fold, which along part of its length is an anticline some 150 ft broad with about 20 ft of closure. The four other known Manto folds are simply structural terraces not more than 60 ft wide, Fig. 2. The axial plane of the Republic Manto fold dips steeply south, but the axial plane attitudes of the other Manto folds are not known. All the known Manto folds lie approximately parallel to the intersections of major Easter faults with the beds, but from 50 to 150 ft away from these intersections. The parallelism suggests that the two kinds of structures are genetically related. The Manto folds are somewhat earlier than the Easter faults, however, since in one case a Manto fold is cut off by a warped portion of an Easter fault.

The Winze folds are much less clearly defined than the Manto folds, and in some cases the presence of a fold is inferred from very limited evidence. The difficulty in outlining Winze folds stems principally from their extreme shallowness; the presence of numerous small faults further complicates the matter. The most well-exposed of the Winze folds is the Northeast Winze fold, an anticline some 300 ft broad with 15 ft of closure, Fig. 3. Of the two other known Winze folds, one is a shallow syncline 150 ft broad and the other an anticline 75 to 150 ft broad with scarcely discernible closure. Where the folds cannot be directly mapped, their presence is inferred from changes of a few degrees in strike and dip of the bedding.

The axial plane attitudes of the Winze folds are unknown. The folds trend approximately parallel to the strike of the Northwester faults, but two of the Winze folds are 500 ft from the nearest Northwester fault, so a genetic relationship is doubtful.

Both the Manto folds and the Winze folds are offset by faults of the Northeaster and Easter groups and are therefore older than those faults. There is no direct evidence as to the ages of the folds relative to the Northwester faults, but probably the folds are older than those faults also. Since the faults are

older than the metamorphism and mineralization, the folds must also be older, a conclusion that is borne out by the fact that nearly all of the ore and some of the metamorphic rocks of the district are localized in shoots lying along the axes of folds of both groups.

Metamorphism and Mineralization

Description of the Rocks: Two kinds of rocks resulted from the metamorphism at Johnson Camp; they are known locally as white tactite and garnetite. A third type of rock, formed during the mineralization phase rather than in the metamorphic phase, is the ore itself.

The white tactite is a very fine-grained greenish-gray to white rock, composed of diopside, quartz, and orthoclase. Diopside is the predominant mineral, making up from 40 to 80 pct of the rock, either quartz or orthoclase or both making up the remainder. Diopside grains are subhedral to euhedral, and always less than 0.5 mm in diam, while quartz and orthoclase form ragged intergrown masses with a maximum dimension of about 5 mm. In a few thin sections, beds of fragmental orthoclase are found.

The garnetite is a coarser-grained reddish rock made up of roughly 50 pct grossularite, 30 pct diopside, and 20 pct quartz and/or calcite. As in the tactite, the diopside is in very small subhedral to euhedral grains, while the quartz and calcite form irregular masses. Much of the grossularite occurs in bands composed of nearly pure grossularite, and in these bands it forms anhedral grains up to 5 mm in diam. Outside these bands, where crystals did not interfere with each others' growth, the grossularite is euhedral. Two types of grossularite, both of them anisotropic, are always present. The older type, which forms the cores of isolated crystals and makes up the bulk of the purer grossularite bands, is dodecahedrally twinned. The younger type, which forms euhedral rims around cores of the older type, is twinned in concentric bands parallel to the rims. This younger grossularite contains a relatively high proportion of manganese.

The ore of the district is markedly banded parallel to the bedding, the sulphides and associated minerals being concentrated in bands from 2 to 14 in. thick, separated by bands of garnetite waste of similar thickness. Aside from scattered spots of sulphides, the garnetite bands are barren. The grade of the sulphide bands is approximately constant, so that the grade of ore broken depends largely on the relative thickness and number of sulphide bands and garnetite bands.

The minerals of the sulphide bands are sphalerite, chalcopryite, tremolite, chlorite, calcite, pyrite, bornite, molybdenite, scheelite, magnetite, and hematite. These minerals everywhere occur together, and in thin section they are seen to be invariably younger than the minerals of the metamorphic rocks. They form a genetic group distinct from the metamorphic minerals. Sphalerite and chalcopryite are the most abundant minerals of the group, making up the bulk of the volume of the sulphide bands. Tremolite, chlorite, and calcite are the most consistently abundant of the gangue minerals; only scattered crystals of the others occur in most of the ore.

Within the sulphide bands, which are more or less continuous over the entire length and breadth of a single orebody, sulphides often form nearly solid masses, with only scattered grains of gangue minerals. The only garnetite minerals present are cor-

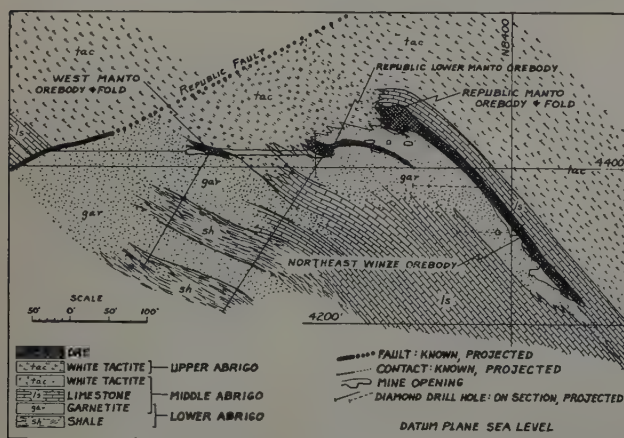


Fig. 2—Cross-section A-A' through Republic mine, showing relationship of metamorphic rocks and ore to structure and stratigraphy.

roded crystals of grossularite and occasional remnants of diopside grains. Since the intervening garnetized beds were so unfavorable for sulphide deposition, it is probable that the beds that now contain the sulphides were at no time very thoroughly garnetized. This conclusion is supported by the fact that at the edges of orebodies sulphide bands often grade out into ungarnetized limestone, while the intervening garnetite bands continue several feet or yards beyond the ends of the sulphide bands.

Outside the orebodies, sulphides occur in scattered small spots in both garnetite and white tactite. For the most part these spots are concentrated in particular beds of the metamorphic rocks, and the bulk of the rocks are barren. In these disseminations, as in the orebodies, the sulphides and their associated minerals are everywhere younger than the metamorphic minerals.

Localization of Ore and Metamorphic Rocks: Stratigraphy was a highly important factor in localizing both the white tactite and the garnetite. Garnetite was formed only in the limestones of the Middle Abrigo and in limy lenses of the Lower Abrigo, while with rare exceptions, white tactite was formed only in the impure dolomites of the Upper Abrigo member and the lower half of the Martin formation. The known exceptions to this latter rule are two areas in which Middle Abrigo limestones, which would normally be garnetized, are tactitized in the footwall of major Easter faults. The most notable such area is in the Republic mine, where a pipe of white tactite 150 ft thick (stratigraphically), 200 ft wide (down the bedding dip), and 1500 ft long lies under the Republic fault, an Easter fault, see Fig. 2.

Within the beds favorable for its formation, the white tactite is much more widely distributed than are the garnetite or the ore in their favorable beds. Throughout the explored part of the district—about 10,000 ft along the strike and 2000 ft along the dip of the beds—most of the beds of the impure dolomites have been altered to very uniform white tactite, regardless of variations in proportions of silica and dolomite from bed to bed in the original sediments. The beds most resistant to the tactitization were the purer quartzites and dolomites. In some areas, particularly above orebodies, even these resistant beds are thoroughly tactitized.

The limestones were much less uniformly metamorphosed. Although the entire thickness of the Middle Abrigo and part of the Lower Abrigo were garnetized around the area of abnormal tactitization below the Republic fault, only the uppermost 75 ft of the Middle Abrigo were consistently favorable for garnetization, Fig. 2. These beds, from the bottom up, are 30 ft of limy sandstone; 20 ft of coarse-grained limestone; and 25 ft of similar limestone with numerous shale beds up to 3 in. thick. The top of the uppermost bed is the contact with the tactitized Upper Abrigo. Usually only the sandstone and the non-shaly limestone are garnetized, but in some disturbed areas the shaly limestone also is garnetized. The beds below the sandstone are rarely altered. Even within these beds most susceptible to garnetization, large areas are essentially unaltered: outside the abnormally metamorphosed areas, nearly all of the garnetite is confined to distinct shoots.

Four garnetite shoots are known in the district: the Northeast winze and 760 garnetite shoots in the Republic mine, Fig. 4, and two others in and near the Moore mine, at the western end of the district.

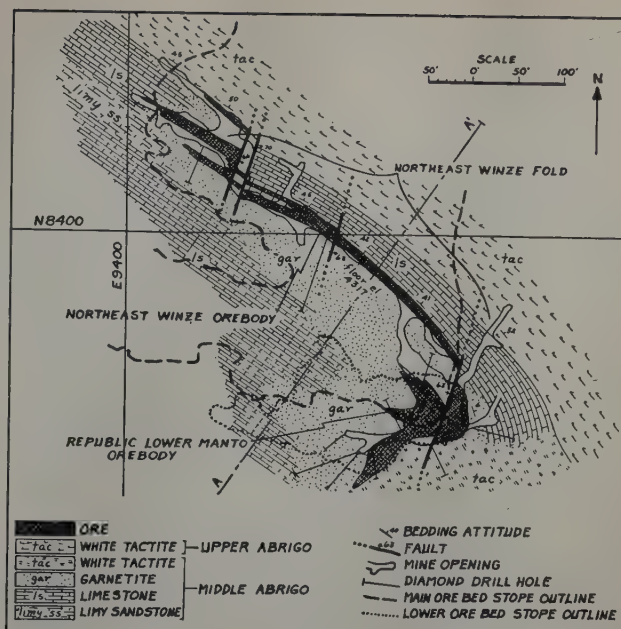


Fig. 3—Geologic map of Republic mine 1100 level, showing ore localized in shallow Winze fold.

Three of these shoots contain ore as well as garnetite. The shoots are at least 500 ft long and from 100 to 300 ft wide. The thickness ranges from 50 ft in places where only the most susceptible beds are garnetized to 75 ft where the overlying limestone as well is garnetized. Within the shoots, most of the rocks are thoroughly garnetized, except some of the purer bands of the sandstone bed and the ore-bearing portions of the shoots, where as much as 50 pct of the rock may be ore. The hanging walls and footwalls of the shoots are bedding planes, and therefore quite regular, but the lateral edges are very irregular in cross-section, since the extent of garnetization along individual thin beds is variable.

The long axes of three of the garnetite shoots trend N 10° W, and in each of these shoots the outline of Winze folds can be discerned locally, but the folds are too shallow to be traced continuously along the entire length of the shoots. The fourth garnetite shoot is known only from diamond drill hole exposures, so no fold can be distinguished. The shoot trends easterly, however, so it is inferred to lie along the axis of a Manto fold.

Besides these four garnetite shoots and the mass of garnetite below the Republic fault, two other masses of garnetite are known in the district, and the presence of several others is indicated. Exposures of these masses are too limited, however, to determine whether they are distinct shoots like those described above or irregular bodies.

The mineralogical composition of the metamorphic rocks, as well as their distribution, demonstrates the importance of stratigraphy in their localization. The white tactite, which is composed largely of diopside, with two exceptions was formed only from dolomitic sediments. The garnetite, composed largely of grossularite with minor amounts of diopside, was formed only from limy sediments. The distribution of the metamorphic rocks within their chemically favorable beds is also in part a reflection of their stratigraphic characteristics as to permeability and competence. The dolomites were evidently highly permeable at the beginning of metamorphism, since they were widely tactitized.

Whether this permeability was due to porosity or to shattering is not known. The limestones, on the other hand, were not generally permeable, but because of their competency they were shattered along zones of structural deformation. The garnetization, therefore, was limited to deformed zones where the metamorphosing solutions could penetrate the rocks.

Structures were evidently of minor importance in localizing the white tactite, except in the cases of the anomalously tactitized areas of the Middle Abrigo. Here faults seem to have acted as barriers, but their exact mechanism and the chemical processes that took place are not known. In the dolomites, where white tactite was the normal result of metamorphism, the only indication of structural control is the presence of areas more intensively tactitized than usual above ore and garnetite shoots.

Both faults and folds localized garnetite masses, and the total volume of garnetite localized by each kind of structure is about the same. However, the garnetite localized by a fault is all in one large mass, beneath the Republic Easter fault, while there are four separate shoots of garnetite localized by folds. In general, therefore, it may be said that folds are more common than faults as localizing structures for garnetite.

For practical purposes the ore, like the metamorphic rocks, can be considered partially localized by stratigraphy: with three small exceptions all the orebodies lie in the uppermost 50 ft of the Middle Abrigo member. The most persistently ore-bearing bed is the non-shaly limestone that is also commonly garnetized, but locally ore extends into the beds above and below.

All the orebodies lie within garnetite masses, the sulphides being interlayered with the uppermost beds of garnetite. The orebodies associated with distinct garnetite shoots such as the Northeast Winze and 760 Winze orebodies of the Republic mine have the same general size and outline in horizontal projection as do their containing garnetite shoots, Fig. 4. The orebodies in the anomalously metamorphosed area of the Republic mine are much smaller than the garnetite mass, but their axes are parallel to the long axis of the area. The three orebodies occurring outside the main ore-bearing beds are in this area. They lie in two lower beds of the Middle Abrigo, but

because the Middle Abrigo is here anomalously tactitized, they are nonetheless in the uppermost part of the garnetite, Fig. 2.

This confinement of ore to the topmost garnetized beds, whatever their stratigraphic position may be, suggests that the top of the garnetite, rather than any particular bed, forms the favorable ore zone. Evidently the mineralizing solutions were active only in the uppermost beds that had been previously garnetized. Throughout most of the district, however, only beds near the top of the Middle Abrigo were garnetized, so that in most instances the ore is somewhat distantly localized by stratigraphy.

Most of the orebodies of the Middle Abrigo are elongate lenses from 12 to 40 ft thick and 35 to 300 ft wide. The length ranges from 150 to 800 ft. The orebodies lie along the axes of either Manto or Winze folds, and their dimensions are largely reflections of the character of the folds containing them. Orebodies lying in the broad shallow Winze folds are wide and thin, while those lying in the narrower, tighter Manto folds are narrow and relatively thick. An exception to this general rule is the West orebody of the Republic mine, which is a broad irregular lens with variable thickness, Fig. 4. Its irregularity probably results from the fact that it lies at the junction of the Republic Manto orebody and the 760 Winze orebody and therefore has some of the characteristics of both, as well as features not found in either.

Courses and Timing of the Mineralizing Solutions: The close spatial relationship between the garnetite bodies and the orebodies demonstrates that the two are closely related genetically, even though the microscopic evidence shows the sulphides to be invariably later than the metamorphic minerals. In all probability, the metamorphic rocks were formed by early solutions, and the orebodies were formed by later solutions from the same source. There was probably no hiatus between the kinds of solutions, but rather a partial overlap. The white tactite was formed during the early stages of solution flow, the metamorphism sealing off the rocks, so that the later mineralizing solutions were unable to penetrate them in any appreciable amount.

The long narrow shape of the garnetite and orebodies, as well as their continuity, suggests that these bodies mark the principal channels in which the solutions flowed. In other words, the ore solutions flowed along the present length of the orebodies and parallel or nearly parallel to the bedding, rather than flowing across the beds and the present orebodies and replacing the favorable beds while leaving the surrounding areas unmineralized. Further evidence for this conclusion is found in variations of the textures of the ores from different parts of the district as described below.

In all the ore, the sphalerite contains numerous blebs of chalcopyrite ranging in size from 0.003 mm to 0.1 mm in diam. The pattern formed by these blebs within individual sphalerite crystals is roughly the same throughout the district: around the edge of each crystal is a narrow zone, some 0.2 mm wide, in which there are no blebs, and within this is a zone about 0.7 mm wide in which blebs are abundant, Fig. 5. In this latter zone, blebs increase in size from the outer edge inward. Variations in the pattern of the blebs appear in the centers of the grains. In the deeper portions of orebodies, the grain centers are nearly filled with large blebs, but in shallower ore the blebs are smaller and less numerous. In some of

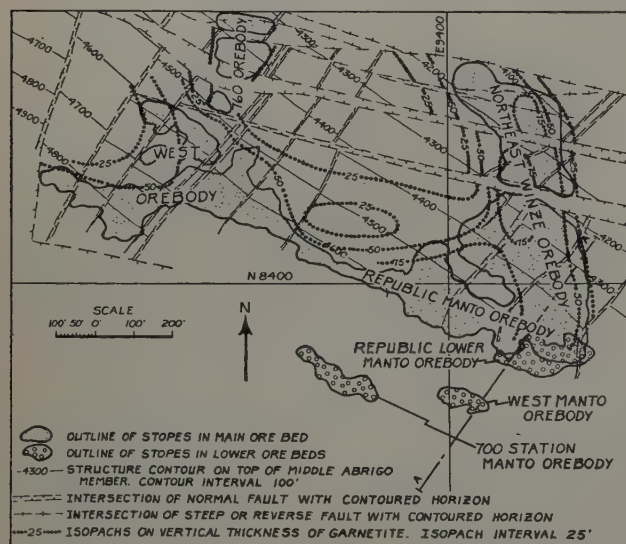


Fig. 4—Republic mine area, structure contour and garnetite isopach map. Outlines of orebodies projected to horizontal plane.



Fig. 5—Relations between sphalerite and chalcopyrite in ore typical of the deeper levels of the Republic mine. X50.

the very near-surface ore, the centers of the sphalerite grains contain no chalcopyrite at all. This change in the pattern of the chalcopyrite blebs is recognizable not only from orebody to orebody, but also along the length of individual orebodies. Thus in the Republic mine, the centers of sphalerite grains from the deepest workings on the Northeast Winze orebody contain numerous large blebs; higher in the same shoot the blebs are smaller. This progressive change does not stop at the top of the Northeast Winze orebody, where the ore takes a sharp change in direction and acquires a new name, the Republic Manto orebody, Fig. 4. Along the length of this orebody the diminution in bleb size continues, showing that the Northeast Winze and Republic Manto orebodies are genetically a single orebody, in spite of the marked change in their shape and direction.

The reason for this change in the character of the chalcopyrite blebs in the sphalerite is unknown. For the present purpose, the important point is that the change is progressive along the length of the orebodies, indicating that the solutions that deposited the sulphides flowed along the present long axes of the orebodies. It is difficult to conceive of any other possible course the solutions might have taken and yet formed the orebodies and caused this progressive change in texture.

On the basis of this evidence, then, the course of the mineralizing solutions must have been approximately as follows. The earliest, metamorphosing solutions reached the exposed area at the bottoms of the Northeast and 760 Winze orebodies and flowed upward from there. Most of the solutions stayed in well-defined channels near the top of the Middle Abrigo member and formed garnetite bodies, but some leaked into the overlying dolomites and altered them to white tactite. The solutions in the principal channels were partially dammed by the Republic fault and formed below it a large mass of

metamorphic rock. It is possible that solutions from another channel, following the Republic fault, were introduced in this area. The later mineralizing solutions followed the same principal channels in their lower parts, forming the Northeast and 760 Winze orebodies by replacing the purer limestone bands that had resisted garnetization. The solutions coming up the Northeast Winze channel were diverted by the mass of metamorphic rock under the Republic fault and flowed westward along the edge of this mass, forming the Republic Manto orebody. Eventually these solutions joined those coming up the 760 Winze channel to form the West orebody. The mixed solutions from the two channels then continued westward, still along the edge of the mass of metamorphic rocks, above the present erosion surface. Probably the three small orebodies within the mass of metamorphic rocks under the Republic fault were formed by solutions that reached the area by some channel not yet located; the sulphide textures of these orebodies indicate that they are not offshoots from the other orebodies.

Conclusions

Stratigraphy was the agent of primary importance in localizing the metamorphic rocks and the ore in the Johnson Camp district: white tactite was formed for the most part only in the impure dolomites, while garnetite and ore were formed only in the limestones. Chemical differences between the dolomites and the limestones were largely responsible for the mineralogical differences between white tactite and garnetite.

Structure was of little importance in the localization of the white tactite; this rock was formed in the favorable beds all through the district, probably because they were quite uniformly, though not highly, permeable. Structures were very important, however, in further localizing the garnetite and ore in certain parts of the limestones. Throughout much of the district, shallow folds shattered beds in the upper part of the Middle Abrigo member enough to make them highly permeable. The metamorphosing and mineralizing solutions used these shattered zones as channelways and formed garnetite and orebodies along them. In rare instances, major faults dammed and diverted the solutions from these fold-axis channels, causing the formation of large masses of metamorphic rocks, but in general faults were of minor importance in localizing both the metamorphic rocks and the ore.

The structural localization of garnetite and ore is to some extent related to stratigraphy as well, in that only certain beds were of the proper competency to be shattered by the folding.

Acknowledgment

The writer is indebted to the Coronado Copper and Zinc Co. for permission to present this paper.

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Mineralization and Hydrothermal Alteration In the Hercules Mine, Burke, Idaho

by Bronson Stringham, F. McIntosh Galbraith, and Garth M. Crosby

THE Hercules mine is located in the northeastern section of the Coeur d'Alene district, approximately 1½ miles north of the town of Burke, Idaho. Surface indications of the ore deposit were first discovered in 1886, but regular mine production was not started until 1902 and was continuous until April 1925, when the known ore had been extracted. Incomplete records show that from 1912 until operations were suspended the mine produced 2½ million tons of ore containing 9.4 pct lead and 7.7 oz of silver per ton, together with an estimated 2 pct zinc, 0.3 pct copper, and 20 pct iron. This operation was the first in a series of mining enterprises culminating in October 1947 with the consolidation of Day Mines, Inc. In the same year it was decided to unwater the levels below the collar of the Hercules shaft in the hope of finding some indication of a recurrence of ore. The unwatering operation has been described in a previous paper.¹ The initial exploration, following recapture of the workings, showed sufficient promise to warrant a detailed study of the mineralogy with modern techniques.

The general geology of the Coeur d'Alene district, including a detailed description of the rock types encountered, has been comprehensively treated by Ransome and Calkins² in their classic paper, and only local background description, therefore, is felt to be appropriate here.

The Hercules deposit transects a portion of the trough of a broad south-trending synclinorium which has been greatly complicated by faulting. More locally, it lies within a block of ground bounded on the east by the O'Neil Gulch fault, a steep north-south overthrust of considerable magnitude, and on the west by a monzonite stock, the outcrop of which is ½ mile or more wide and 5 miles long. The country rock is composed of thin to medium-bedded argillites and argillaceous quartzites of the Prichard and Burke formations, the oldest members of the Pre-cambrian Belt Series of sediments in the area, believed to be of Algonkian age. The contact between them is a conformable gradation. The argillite is colored gray to tannish-gray and is fine-grained, compact, and generally massive in structure. Under the microscope the unaltered argillite is seen to be composed principally of anhedral quartz and a few feldspar grains which were at one time presumably partly rounded sand grains, but as a result of recrystallization and cementation by silica, the interstices are now almost obliterated and quartz grains show crenulate boundaries. The sizes of these crystals vary from 0.5 mm down to 0.1 mm in greatest dimension. In all specimens sericite comprises 10 to 20 pct of the

rock and is present abundantly between most of the grains as flakes or shreds which vary considerably in size. Sometimes they form a fine felt-like mat or aggregate, and sometimes flakes are seen which appear to be good muscovite. In some specimens, separated rhombic-shaped carbonate grains are abundant, and in some instances these have been changed to sericite.

Mining operations to date have explored the Hercules vein to a maximum vertical depth of 3600 ft below its outcrop, and along a maximum strike-length of 3600 ft on certain of the lower mine levels. The main orebody is irregular in outline, extending over a variable strike-length of 400 to 1500 ft; and it is intersected by a strong transverse fault that has been traced from the surface to the bottom level. This has been named the Hercules fault, and apart from the vein itself, it is the most prominent structural feature in the mine. There is good evidence that it existed prior to the introduction of ore solutions and may have influenced ore deposition, but it was also the locus of important post-ore displacement and shows a progressive right-handed horizontal component reaching 200 ft on the deeper levels. Its vertical component is not definitely known but may be considerably greater. The fault strikes 20° N to 50° E and dips westerly at angles of 70° to 45°, flattening in dip where it crosses the original orebody from east to west between 1000 and 1600 ft below the surface. At about 3000 ft in depth the Hercules fault is joined by a vertical fault of similar strike, and the major post-ore displacement below their junction is taken up along this vertical branch of the structure, now called the Mercury fault. Recent work has been concentrated in this vicinity. Another structural feature of special geologic interest, though of little economic importance, is the occurrence of a porphyritic dike in this area. This lies a short distance above the Hercules fault, essentially parallel to it, and is 5 to 15 ft in thickness. It appears at first glance to cut the mineralization, suggesting *push-apart* relationship, but small stringers of the vein minerals have been observed to penetrate the dike for a matter of inches at several points. The dike is thought to be related to the monzonite intrusion.

A vertical longitudinal projection of the mine is shown in Fig. 1, which illustrates most of the features discussed above.

The Hercules vein was deposited along the course of a strong, persistent shear zone that now appears as a braided network of gouge seams running through more or less crushed and shattered country rock. It strikes 70° N to 80° W and dips southerly at an average of 75°. Barren parts of the structure vary in width from less than 1 ft to more than 15 ft. The width of mineralized segments may be double that. Although the evidence is not conclusive, pre-mineral, normal movement along the zone may be 1000 or 1500 ft. The horizontal component is unknown. Post-ore movement appears to have been

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Discussion on this paper, TP 36881, may be sent (2 copies) to AIME before Feb. 28, 1954. Manuscript, Dec. 31, 1952. New York Meeting, February 1952.

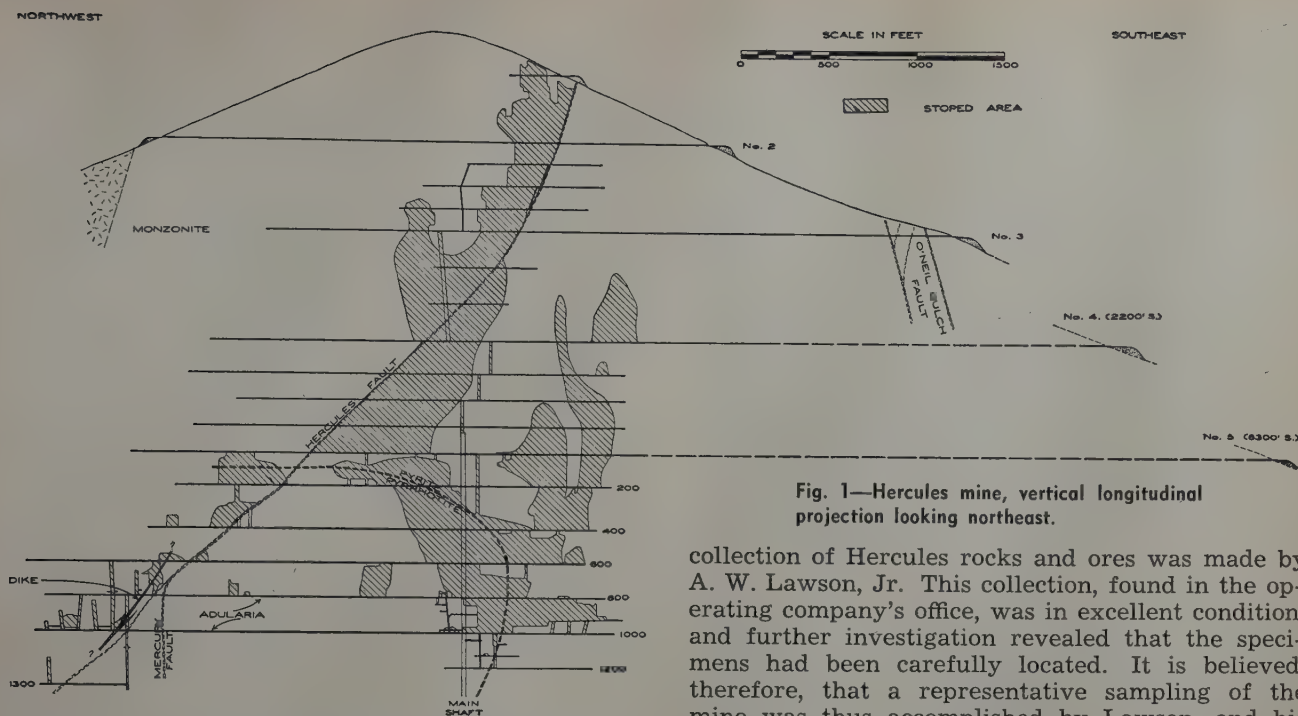


Fig. 1—Hercules mine, vertical longitudinal projection looking northeast.

insignificant, although moderate shearing can frequently be observed on the walls of the vein, more commonly on the footwall, resulting in a well defined slip. The common vein pattern shows a principal strand of massive mineralization accompanied by a variable width of parallel or branching stringers in either or both walls, so that the total width of mineralization is not uncommonly twice that of the main vein.

The mineral assemblage is particularly interesting and includes silicates, carbonates, oxides, and sulphides. The principal gangue minerals are pyrrhotite, magnetite, siderite, grunerite, and quartz with subordinate amounts of pyrite, calcite, biotite, garnet, and chlorite. The ore minerals are galena and sphalerite, with sporadic minor occurrences of chalcopryrite. Although the high-grade ore appears most commonly as irregular masses within the vein, a poorly developed banding of minerals can usually be seen, parallel or at low angles to the vein walls. The mineral textures vary from dense to coarsely crystalline with little apparent relation to vein widths, and one rather suggestive texture is a peculiar alignment of crystals simulating gneissic structure that is occasionally noted in the coarse galena. The vein mineralization as a whole is believed to have been deposited predominantly by filling, replacement playing a subordinate role.

Development of Hydrothermal Minerals in or Near the Hercules Fissure

Studies of the hydrothermal history of the minerals in or near the Hercules fissure began in the summer of 1950 when nearly 100 specimens were collected from the mine for detailed observation. Thin sections and polished sections were prepared of most of the specimens, and during the late summer and fall of the same year the mine was revisited and laboratory investigations were completed. The open lower levels of the mine were examined in considerable detail. Some of the older upper levels were also inspected, but because of their inaccessibility, detailed observations on these levels were not possible. During the 1920's, when the mine was in full operation and most of the levels were open, a

collection of Hercules rocks and ores was made by A. W. Lawson, Jr. This collection, found in the operating company's office, was in excellent condition, and further investigation revealed that the specimens had been carefully located. It is believed, therefore, that a representative sampling of the mine was thus accomplished by Lawson, and his samples were freely examined.

Openings in the mine are largely confined to the fissure proper, and the scarcity of crosscuts available for collection made the study of the rocks transverse to the fissure somewhat limited.

Biotite: A very fine-grained, massive-appearing, black to dark green biotite is the most important and widespread of the minerals formed by hydrothermal alteration and is present throughout the entire mine in or near the main Hercules fissure zone. In a broad general way it is more intensely developed near ore shoots but small amounts are observed in wall rock, at least as much as 15 to 25 ft from the main vein on the lower western levels.

X-rays, differential thermal analyses, and optical properties prove distinctly that this mineral is true biotite. Where it makes up practically 100 pct of the rock, it is very massive in appearance and is colored black to greenish black. It is also very soft, and where tectonic movement has occurred this mineral usually has taken up the slippage and often shows prominent shiny slickensided surfaces. Near the Hercules vein the mineral may appear as a dark band, up to 7 cm wide, in gray argillite, where soaking on either side of a solution channel has allowed the development of a considerable thickness of biotite. When abundant it appears under the microscope as a solid mat of randomly oriented flakes, but if it constitutes only a small portion of the rock it is usually distributed in isolated or connected patches of aggregates or sometimes as distinct veinlets, see Fig. 2.

Biotite is the first hydrothermal mineral to form in the Hercules vein. All the other minerals are found to occur in veins traversing rock containing biotite.

Andradite: Andradite in small individual crystals occurs primarily in the lower western levels of the mine and is particularly prominent in the Hercules fissure zone, but it is also noted in small amounts in wall rock as much as 15 ft from the main vein.

Quartz: Quartz is present in most of the specimens of vein material. There seems to be no evidence to determine that there were separate distinct periods when quartz was formed, and it is

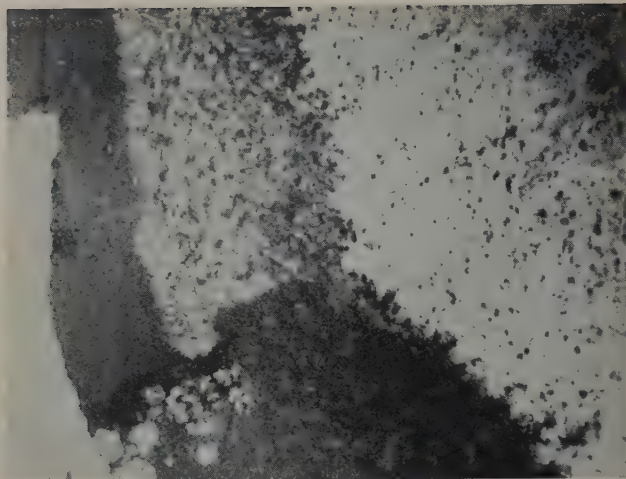


Fig. 2—Biotite from the Hercules mine as revealed under the microscope. Biotite is the first hydrothermal mineral to form in the Hercules vein.

believed that quartz was probably being deposited during all stages of the mineralization.

Grunerite: Grunerite occurs principally in the main Hercules fissure zone in the western part of the vein on the intermediate and lower levels. It is never found in the wall rock for any great distance on either side of the fissure. It occurs in a form which is distinctly fibrous to prismatic with some crystals as much as 1.5 mm long. The color of the mineral is a dark grayish-tan in practically all cases. Grunerite veins cut biotite, andradite, and glassy quartz, but magnetite and sulphides are found to traverse grunerite, thus establishing the relative age of grunerite as premagnetite and presulphide.

Adularia: Adularia was discovered microscopically in two specimens taken from the main Hercules fissure zone between two stoped areas where essentially no prominent sulphide mineralization was present, see Fig. 1.

Chlorite: The chlorite occurring principally in argillite close to the Hercules fissure appears generally green. It is rarely massive, occurring principally as disseminated flakes and patches in otherwise unaltered rock.

Siderite: Light gray to tan siderite is found throughout the mine on the Hercules fissure not only where ore is present but also in many places where ore is absent. It may occur by itself or associated with magnetite and sulphides; thus there seems to be no fixed spatial relationship between the occurrences of ore and siderite.

Also, scattered small crystals of siderite, 0.1 to 0.2 mm, are often found in wall rock adjacent to the main vein. It was either wholly introduced into the wall rock or some of the iron which was introduced into the vein system spread into the wall rock and replaced the calcium in the original calcite of the argillite and converted it to siderite.

Magnetite: Massive to granular magnetite is present in the Hercules fissure throughout the entire mine and nearly always accompanies ore, but it may also occur in places where ore is not present. It seems slightly more abundant in the lower levels.

Sulphides: All the sulphides found during this study, that is, pyrrhotite, pyrite, arsenopyrite, jamesonite, chalcopyrite, galena, and sphalerite, were identified by conventional optical, etch, X-ray, and microchemical methods. It is not necessary to describe the well-known properties of each of these

minerals. No chemical analysis was made of the pyrrhotite to determine the precise iron and sulphur ratio, but it was found to be slightly magnetic.*

* Four additional minerals, agentite, cubanite, niccolite, and tetrahydrite, have been reported as occurring in the Hercules by earlier observers (private reports).

Calcite and Late Pyrite: Calcite veins, 1 to 10 cm wide, trending parallel to the main fissure were found cutting through sulphides in several instances. A very few small crystals of pyrite were discovered to occur in vugs and open spaces on the calcite. Also, some veinlets of pyrite, 2 mm wide, were found in sphalerite. These two occurrences of pyrite are believed to have been formed at the same stage of mineralization. It is also believed that they represent the last hydrothermal activity in the Hercules fissure.

From the information obtained by an examination of the specimens of both A. W. Lawson's collection and the present one, it was found that the following spatial relationships, in general, hold true. Biotite, chlorite, magnetite, siderite, quartz, chalcopyrite, galena, and sphalerite are found in the Hercules fissure on practically all levels of the mine. Only biotite, andradite, chlorite and siderite also occur in the wall rock adjacent to the fissure. Andradite and grunerite seem to be more or less restricted to the lower western levels. The distribution of early pyrite and pyrrhotite is one of the most interesting problems found in the mine, see Fig. 1. Pyrrhotite and pyrite are rarely together in the same specimen, and in the lower western levels, pyrrhotite is the exclusive iron sulphide, whereas in the eastern and upper part of the mine, pyrite is essentially the only iron sulphide. There seems to be a narrow transition zone where pyrite and pyrrhotite may occur together, and as far as can be determined, it runs from at least No. 5 level, west of the shaft, to slightly east of the shaft at the 1200 level. Pyrite is also found in the cores of drill holes which encounter the vein considerably below the 1200 level. This suggests that the transition zone between pyrrhotite and pyrite may extend downward steeply to the west from the 1200 level. The meaning of this distribution of the two iron sulphides is not at all clear. The possibility is suggested that the western part of the orebody was deposited at a somewhat higher temperature, thus pyrrhotite, being considered a higher temperature mineral, was formed. Arsenopyrite and jamesonite are found only on the edges of stopes in the upper eastern portions of the mine. Adularia occurs on the 800 and 1000 levels in a completely barren section of the Hercules fissure, see Fig. 1.

In a review of the mineralization of the Hercules fissure, disregarding quartz, which seems to have formed continuously throughout the mineralizing period, a very interesting and perhaps significant grouping of the minerals is possible according to the time or period of development, and at least five or perhaps six distinct stages could be postulated as follows, see Fig. 3:

Stage 1. Silicate: In the first stage, the minerals biotite, andradite, grunerite, adularia, and chlorite were formed. During this period much Fe_2O_3 , FeO , and SiO_2 were introduced along with small amounts of MgO , Al_2O_3 , and a very little K_2O and CaO . Whether or not some or all of these materials are from a magmatic source cannot be positively determined, since the unaltered wall rock contains all of these substances in various amounts and could have

possibly contributed some constituents to the primary solutions as they traversed the rock.

Stage 2. Carbonate (Siderite): The second stage may be distinguished from Stage 1 on the basis of the different nature of the radical. The concentration of FeO and/or Fe₂O₃ in the introduced solutions continued to be high, but in this instance CO₂ or possibly CO were also available and the mineral siderite was deposited.

Stage 3. Oxide (Magnetite): This stage could be separated from the second on the basis that the radical is different. Here it may be supposed that Fe₂O₃ and FeO were the only materials deposited. Of course, it may be reasoned that Stage 3 (magnetite) and Stage 2 (siderite) might represent the same stage but that during Stage 3 there was an excess supply of iron over CO₂ or CO to make siderite, and magnetite developed as a result. The nature of the distribution of these two minerals, however, might serve to refute this idea to some extent since siderite is frequently found in many areas by itself, and often magnetite occurs without being associated with siderite.

Stage 4. Sulphide: In this most important stage the sulphides pyrrhotite, pyrite, chalcopryrite, arsenopyrite, jamesonite, galena, and sphalerite were formed. This mineral assemblage indicates that sulphur was a profuse constituent in the solutions at this time and that Fe, Pb, and Zn were abundant together with a little Cu, As, and Sb. Up to this stage, Fe was present constantly in the depositing solutions, but the supply was apparently discontinued during the development of the last three minerals, jamesonite, galena, and sphalerite.

On this basis, the sulphide stage could be separated into two separate substages, one where iron was deposited and the other where iron was not deposited. The paragenetic relations between the galena and sphalerite stage and the earlier sulphides, excepting jamesonite and arsenopyrite, quite clearly indicate that there is a distinct separation in time of deposition between the two stages. It is therefore considered reasonable to separate the sulphide stage into two sub-stages.

Stages 5 and 6. Calcite and Pyrite: These last two stages are distinctly later than the earlier stages as shown by both structural and compositional evidence. It could be argued that the deposition of calcite in crosscutting late veins represents a supergene action, and there is really no reason to refute this, except that the presence of pyrite crystals in vugs in the calcite would not be so easily explained by the same process. Pyrite could form from supergene waters if they had been in contact with a great deal of sulphide material, resulting in a reducing environment. However, the hypothesis that the calcite and small amounts of pyrite formed during late phases of hydrothermal activity seems to be the most reasonable. The possibility that the calcite was supergene and that the later pyrite was deposited by a weak resurgence of hydrothermal activity seems remote.

Hydrothermal Alteration

If hydrothermal alteration is thought of in the conventional sense, i.e., mineralogical changes in the wall rock near an orebody, there is a surprisingly small amount of it in the Hercules area in relation to the size of the deposit as contrasted with that found at Butte,³ Santa Rita,⁴ Tintic,⁵ Bingham,⁶ and other districts where this feature has been studied. As far as can be observed, the wall rock adjacent to the

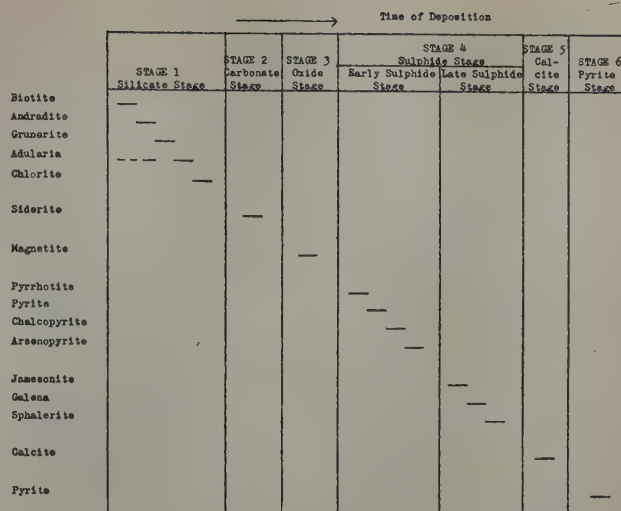


Fig. 3—Paragenesis of hydrothermal mineral development in the Hercules vein.

Hercules fissure has been affected by the development of biotite, andradite, and chlorite for only a short distance, 15 ft maximum, on each side of the mineralized fissure and this condition is found only in the lower western levels. Near heavy sulphide mineralization on the lower eastern levels, many instances were noted where essentially fresh rock was adjacent to the sulphides, and on the upper levels this condition definitely prevailed. Hydrothermal siderite was found in small amounts within the wall rock for a distance of a few feet away from the fissure and adularia is present only in the barren sections of the Hercules fissure proper.

From the highest level of the mine to the deepest, biotite, chlorite, and siderite are present in the wall rock at various points within and very near the main fissure. These minerals are not always associated with the ore and may occur in barren zones. Ore is also often seen without appreciable green sericite or chlorite associated with it.

Four possible reasons are presented to explain the lack of widespread hydrothermal effects in the Hercules. 1—The channels within the fissure were sufficiently open at the time of early solution movement to allow them free and easy movement up through the fissure, and pressures were not high enough to force solutions into the wall rock. 2—The wall rock was not sufficiently fractured or porous to allow much solution penetration into it even though pressures may have been high. 3—The supply or amount of altering solution was so limited that a large area could not be affected. 4—The time that the existing altering solutions were present was so short that diffusion into the wall rock was limited.

The physical and chemical environments under which biotite and chlorite form are not well known, and an hypothesis regarding the pH of the altering solutions forming these two minerals is therefore not possible. Siderite is reasoned to form under alkaline conditions. Sericite is known to form under acid conditions at high temperatures, above 350°C, but at lower temperatures, below 350°C, it forms under alkaline conditions.⁷ If biotite forms under conditions similar to sericite, interpolation of the temperature and pH of the solutions is possible with the assumption that the pH of the solutions did not appreciably change during the first (silicate) and second (siderite) stages; then in both stages the solutions were alkaline and the temperatures were low, below

350°C. This conclusion, which is necessarily based on an assumption, brings forcefully to the student of hydrothermal alteration that more laboratory data are very much needed regarding the physical-chemical conditions under which minerals form, so that even such a restricted problem as the pH of hydrothermal solutions may be better interpreted.

The restricted nature of the green sericitic and sideritic alteration in the Hercules mine makes the use of hydrothermal alteration poor as a guide in exploration. These alteration minerals do not form a much larger target than the orebody itself and in some places the orebody is present where these alteration minerals are entirely absent. It may be stated, however, that if in crosscutting or drifting any of these minerals were found in considerable amounts without associated ore, it would be presumed to be an encouraging situation.

The widespread sericitic alteration of the rocks in the Coeur d'Alene region has been known for some time and is locally referred to as *bleaching*. Hoyt S. Gale first recognized this alteration, but he noted it only in private reports. Shenon and McConnel⁸ recorded their observations on the sericitic type of alteration in 1939. It is understood that a paper by Thomas Mitchem as a Ph. D. thesis from Columbia University dealing in detail with this alteration is in press. A detailed account of it is therefore not appropriate here, but the relation to the overall district conditions may be of some interest. Quite a number of sericitic type of rocks were collected and studied. All the formations exposed in the Coeur d'Alene district have at some place or another been partially or completely altered to sericite. A description of the field occurrence of this alteration may best be quoted from Shenon and McConnel where they mention that within the district "several well defined zones of hydrothermal alteration, as much as one-half mile wide, are found. . . . The rocks in the zones regardless of their original color or

composition are sericitized and are altered in color from gray or purple to some shade of green In general, the argillaceous rocks have been most completely altered but where the alteration processes were intense, massive quartzites, as well as argillites, have been altered to sericitic rock." It is of particular interest that no alteration of the above type is found anywhere in the vicinity of the Hercules fissure. The presumption is, then, that although the sericitic type of alteration is associated with ore in the district in a very broad general way, ore shoots do not always follow it and bleaching therefore cannot be regarded as a positive guide to ore.

Acknowledgments

The authors wish to offer thanks and appreciation to Henry L. Day, President of Day Mines, Inc., who gave his full support and encouragement to this work and offered valuable advice and information during the course of the investigation.

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Starches and Starch Products as Depressants In Amine Flotation of Iron Ore

by C. S. Chang, S. R. B. Cooke, and R. O. Huch

IN the flotation of iron ores laurylamine derivatives have been used for considerable time.^{1,2} To effect satisfactory separation of the gangue, predominantly silica, from the iron oxide minerals, the addition of specific depressants for the iron oxides has been found necessary. Starches and starch products have been used for this purpose.³ There is, however, very little published information on the effects of these materials and other variables on this

process. In some cases it has been found that prior acid scrubbing and desliming are essential,⁴ and depending upon the efficiency of the subsequent washing step, there will be accompanying changes in pH.

It was the object of this investigation to test and observe the effects of the above factor as well as to examine and compare the action of various starches and starch products.

Ore Preparation

The ore used in this experiment was a wash-ore tailing obtained from the western Mesabi range. Microscopic examination and x-ray analysis of the ore showed it to consist of hematite, goethite, and quartz. Physical properties of the iron minerals ranged from hard and crystalline to earthy and ocherous. The quartz grains were discolored either because of iron-oxide inclusions or because of a surface coating of iron oxide. The ore assayed 27.52

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Discussion on this paper, TP 3611B, may be sent (2 copies) to AIME before Feb. 28, 1954. New York Meeting, February 1954.

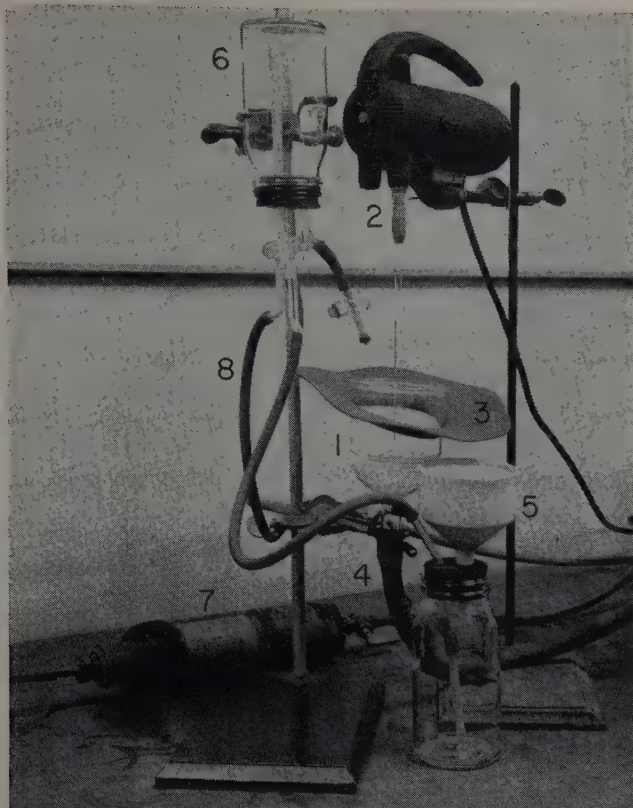


Fig. 1—The flotation cell with auxiliary equipment. (1) Buechner fritted glass funnel, (2) stirrer with pyrex stirring rod, (3) rubber sleeve, (4) air inlet, (5) Buechner funnel, (6) solution reservoir, (7) air filter, and (8) vacuum line.

pct iron and 57.11 pct silica, and, as received, had been screened through 20 mesh.

To prevent overgrinding, the -100 mesh material was first removed by screening. The +100 portion was ground in a Denver laboratory rod mill until all of it passed through 100 mesh. The ore was then deslimed at roughly 20 microns in a 3-ft Denver bowl thickener in closed circuit with itself.

Acid scrubbing was accomplished by placing 1 kg of deslimed ore in a laboratory Fagergren cell, enough water to bring the pulp level to the top of the baffle plate, and 1 ml of concentrated sulphuric acid, equivalent to 3.6 lb per ton of ore. The pulp at this stage had a pH of 2.1. (All reagent additions used in the paper refer to the short ton.) The pulp was agitated for 5 min. Because of the large quantity of fresh slime material produced during the acid scrubbing, the ore was again deslimed, by sedimentation, until the supernatant liquid became clear. This also removed substantially all the sulphuric acid. The ore was given a rinse with demineralized water before it was dried. It was then thoroughly mixed in a box-type tumbler, split with a Jones splitter, and stored in glass bottles. The flotation feed represented approximately 80 pct of the original ore. A screen analysis is given in Table I.

The specific surface of the prepared ore, determined by the krypton gas-adsorption method,⁵ was 1.82 m² per gm. The material was fractionated in acetylene tetrabromide at specific gravity 2.96. The float, 47.9 pct by weight of the material, had a specific surface of 0.22 m² per g and was essentially quartz. The sink product carried practically all the iron minerals as well as some locked quartz. By calculation, the sink product had a specific surface of 3.28 m² per g, equivalent to 94 pct of the total surface of the flotation feed.

To avoid vigorous agitation, which always resulted in the production of fresh slimes, a test cell, fundamentally of pneumatic type, was devised. This cell consisted of a 350-ml capacity Buechner fritted glass funnel with medium porosity frit. Air was admitted through the bottom of the cell and controlled by a needle valve. Admission of air alone did not give sufficient agitation to keep all the material in suspension; hence supplemental mild agitation, provided by a motor-driven glass stirring rod, was used

Table I. Screen Analysis of Flotation Feed

Screen Size, Mesh	Wt, Pct	Iron, Pct
-100 +150	37.29	29.23
-150 +200	40.28	29.62
-200 +270	17.45	41.48
-270 +400	3.63	61.43
-400	1.35	66.65
	100.00	33.20

in the cell. To facilitate the collection of the froth, a rubber sleeve was fitted to the neck of the cell.

The complete assemblage is shown in Fig. 1. The froth flows from the cell into a Buechner funnel where the froth is filtered and the filtrate transferred to the solution reservoir by suction. This solution is then available for re-introduction to the cell when desired and eliminates the necessity of adding water during a test.⁶

To the cell were added 50 g of ore and 200 ml of solution containing those starches and starch products required for the test. This mixture was conditioned, with agitation only, for 2 min, after which the laurylamine acetate collector was added. This was followed by an additional 30 sec of conditioning. The air was then admitted to the cell and the froth was removed until it was barren of mineral.

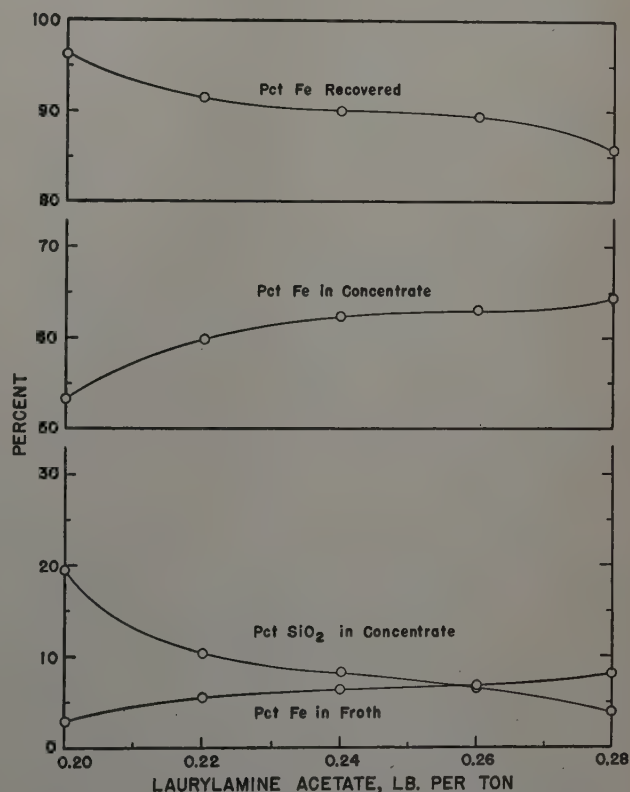


Fig. 2—Effect of collector addition with 0.03 lb Gum 3502 per ton.

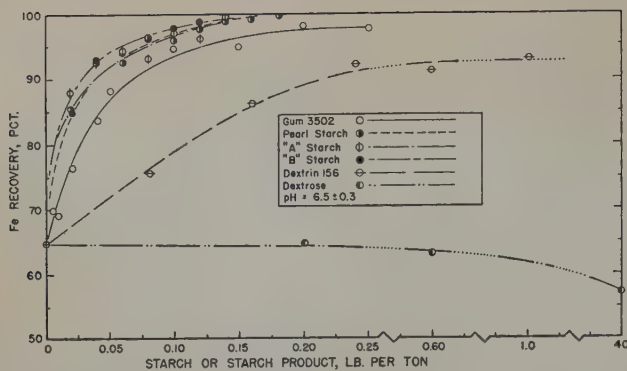


Fig. 3—Effect of starch addition on percent iron recovered. Laurylamine acetate addition at 0.26 lb per ton.

This removal required 1 to 3 min depending on the froth characteristics. The froth contained most of the silica, and the iron concentrate remained in the cell. Assays were made on these two products.

At the end of each test the pH of the pulp liquid was measured with a Beckman Model G pH meter. In all cases thorough washing and desliming followed the acid scrubbing, and the pulp reached an equilibrium pH of 6.5 ± 0.3 . All tests in which pH was not regulated were made within this range.

Before being used in any subsequent tests, the flotation cell was thoroughly washed with hot water and rinsed with demineralized water.

Collector: The laurylamine acetate, 99.9 pct pure, was supplied by the Research Division of Armour and Co. For addition to the cell, a freshly prepared 0.1 pct solution was used.

Depressants: All starches and starch products tested, except the dextrose, were obtained from Corn Products Refining Co., New York. They are described below.

Gum 3502 is a water-soluble, partially degraded starch. The material is readily soluble in hot water.

Pearl starch is corn starch which has been prepared with a minimum of alteration. It is essentially insoluble in water at room temperature. The solution added for the test was prepared by digesting it in water in an autoclave at 120°C for 1 hr.

A and B starches are fractions of corn starch. The A fraction, which is quite soluble in water, consists mainly of amylose, i.e., unbranched and long-chain starch molecules. The partially water-soluble B fraction consists primarily of amylopectin, i.e., branched starch molecules. The preparation of solutions of these starches was similar to that for pearl starch. However, before treatment in an autoclave, it was found necessary first to disperse them in benzene and later to remove this by adding hot water and boiling.

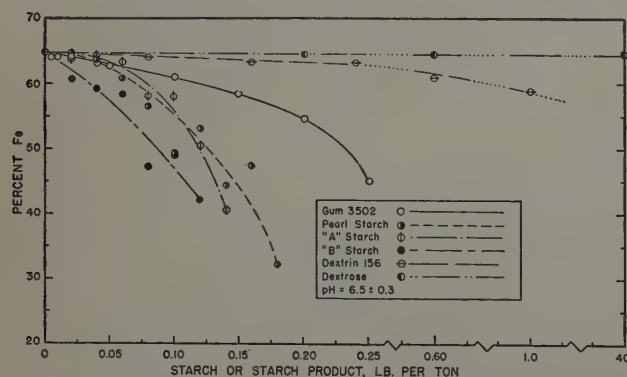


Fig. 4—Effect of starch addition on percent iron in concentrate. Laurylamine acetate addition at 0.26 lb per ton.

Dextrin 156, a decomposition product obtained by dry heating of starch, is wholly water-soluble.

Dextrose is a monosaccharide resulting from complete degradation of starch. The material used was of C.P. reagent grade.

All the starches and starch products were added to the flotation cell as 0.1 pct solutions.

Other Reagents: The pH regulators, sodium hydroxide and hydrochloric acid, were of analytical reagent grade. Demineralized water, containing less than 0.1 ppm of salts as NaCl, was used in the preparation of all solutions and for all test work.

Effects of Collector Addition: It was necessary to first decide on a collector concentration at which the tests should be run. Lindroos⁴ found that for the same ore a fairly satisfactory separation was obtained with 0.03 lb of Gum 3502 per ton. A series of tests was therefore made by using this same gum concentration and varying the quantity of collector. The results are shown in Fig. 2.

It can be seen that the iron recovery decreases and the iron content of the concentrate increases with increasing collector addition. In fact, preliminary tests showed that the whole ore could be floated if an exceedingly large quantity of collector were used. However, when excess starch was added, the ore was completely depressed and it could not be made to respond to flotation again even with a large addition of collector.

At a collector addition of 0.26 lb per ton, a good grade of concentrate was obtained assaying 63 pct iron with only 6 pct silica representing a fairly high recovery of 89 pct. At lower collector additions, fair results were obtained, but the froth possessed poor characteristics and it was necessary to add some frother to remove completely all the floatable minerals. Since introduction of an additional variable was undesirable, the value of 0.26 lb of laurylamine acetate per ton was chosen for all subsequent tests.

Effects of Various Starches and Starch Products: The starches and related products used in these tests have already been listed. The amount of collector was kept at 0.26 lb per ton while the starch addition was varied. The results are given in Figs. 3, 4, and 5.

In interpreting the results, it will be of assistance to remember that the flotation of the iron ore at a pH of 6.5 with no reagent addition other than 0.26 lb of laurylamine acetate per ton gave an iron recovery of 65 pct in a concentrate assaying 64.6 pct iron and 3.0 pct silica. The effect of an ideally selective depressant for iron minerals would be to maintain a grade of concentrate reasonably close to that given above, with the overall iron recovery simultaneously approaching 100 pct. No such depressant is known, but the relative efficacy of the reagents tested as iron oxide depressants is measured by the positions of their respective curves in the three figures. It should also be remembered that technologically and economically the iron concentrate may contain as low as about 61 pct iron and about 8.5 pct silica. Iron recovery, on the other hand, depends entirely upon economic considerations which need not be discussed here.

In Fig. 3, the recovery of the iron in the concentrates is plotted against the iron depressant added. Fig. 4 shows the relationship between the grade of concentrate, expressed as percent iron, and the amount of depressant added, and Fig. 5 shows the iron content of the tailings, plotted against the amount of starches or starch products added.

In these graphs it will be seen that the general trend, with the increasing depressant additions, is

toward a higher recovery of iron (Fig. 3), a lower grade of iron concentrate (Fig. 4), and less iron reporting in the froth (Fig. 5). This is to be expected, since, with the increasing starch additions, a greater depressing effect is exerted, not only toward the iron minerals, but also toward the quartz grains. At the low depressant additions, there is evidently insufficient starch to coat all of the iron minerals present. As a result, some of these minerals are sufficiently coated with the collector to be floated.

It is obvious from all these figures that the B-fraction starch is the least selective depressant. This is indicated by the rapid decrease in the grade of iron concentrate and by the steeply rising iron recovery curve with increase of reagent. It is concluded from these results that of any reagents tested the B-fraction starch is the least selective depressant at a pH of about 6.5.

The effects of Pearl starch, A-fraction starch, and Gum 3502 are similar to each other. The grade of concentrate produced by them is superior to that given by the B-fraction, although total iron recoveries are approximately the same with the exception of Gum 3502, which gives slightly less recovery than the others.

Dextrin permits production of a very good grade of concentrate, but the percent iron recovery is less than with any other starches or starch products but dextrose, which has no effect on flotation operation.

Both Pearl starch and the B-fraction starch contain water-insoluble constituents, and these could possibly have an effect on flotation. To clarify this point, the starch solutions were centrifuged to remove the insoluble materials. Flotation tests using these *cleaned* starches gave results no different from those using the whole starch products.

Effects of pH: In the investigation of the effects of pH, collector and starch concentrations were kept constant while the pH was varied from 3.9 to 11.2. The collector addition was maintained at 0.26 lb per ton with Gum 3502 at 0.10 lb per ton in one set of tests and Pearl starch at 0.06 lb per ton in another. A series of tests was also made with no depressant used. The results are shown in Figs. 6 to 8.

There is a decrease in the iron recovery in near-neutral pulps, Fig. 6. This decrease is minor in the presence of either starch or gum but is very pronounced when neither is used. A corresponding increase in percent iron in the concentrate, Fig. 7, is shown in the same region for all of them, with the grade of concentrate dropping off at high and low pH. At the concentration of laurylamine acetate employed, a precipitate, presumably laurylamine hydroxide, begins to form in pulps of higher pH than 9. Since this is the case, the reason for the poor grade of the concentrate at high pH, with resulting high recovery, is probably that insufficient laurylamine ion is available for collection and therefore more silica and iron minerals remain in the cell. The low grade of iron concentrates obtained at low pH may be attributed to insufficient adsorption of the collector ion on the minerals. Hydrogen ion, in acid pulps, may compete strongly with the laurylamine ion for occupancy on the mineral surfaces.

Particle Size in Flotation Products: In an attempt to explain the appearance of quartz in the concentrate and iron minerals in the froth, screen analyses were made on flotation products of two tests, one in which 0.03 lb Gum 3502 per ton was used and one in which 0.04 lb Pearl starch per ton was used.

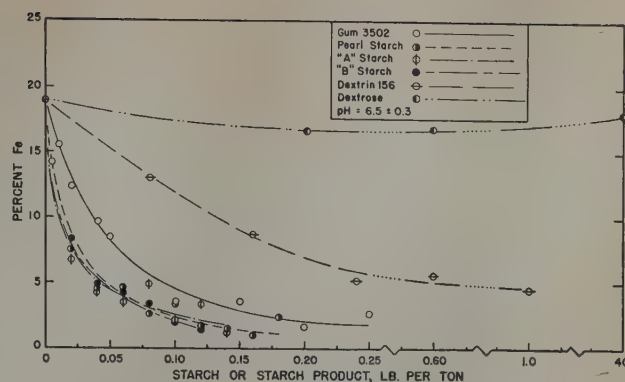


Fig. 5—Effect of starch addition on percent iron in froth. Laurylamine acetate addition at 0.26 lb per ton.

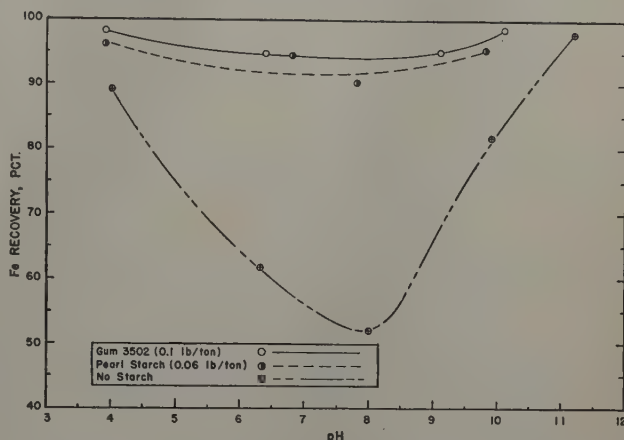


Fig. 6—Effect of pH on percent iron recovered, with no starch, with Gum 3502, and with Pearl starch. Laurylamine acetate addition at 0.26 lb per ton.

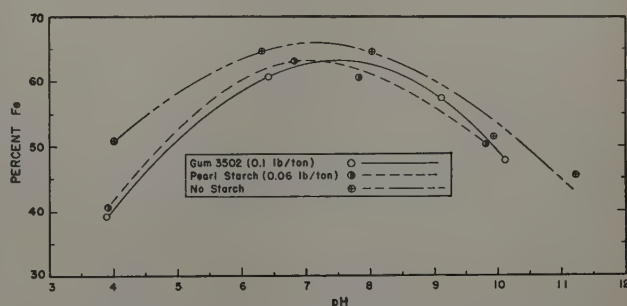


Fig. 7—Effect of pH on percent iron in concentrate, with no starch, with Gum 3502, and with Pearl starch. Laurylamine acetate addition at 0.26 lb per ton.

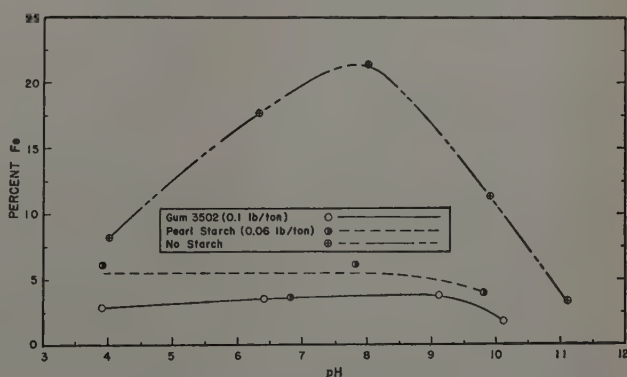


Fig. 8—Effect of pH on percent iron in froth, with no starch, with Gum 3502, and with Pearl starch. Laurylamine acetate addition at 0.26 lb per ton.

Assays were made on the screened products and the results are given in Tables II and III.

These tables show that most of the quartz in the concentrate is contained in the coarsest screen fractions. Microscopic examination of these fractions showed that most of the quartz appeared as free grains which apparently failed to respond to collection. In the froth product, the percent iron rises very sharply as the particle size decreases, indicating that a large part of the iron consists of free oxide particles which presumably are carried mechanically into the froth. Microscopic examination

Table II. Screen Analyses of Flotation Products with Gum 3502, 0.03 Lb per Ton

Screen Size, Mesh	Wt, Pct	Iron, Pct
Concentrate, 46.7 pct by weight		
—100 + 150	37.18	58.95
—150 + 200	36.86	63.54
—200 + 270	19.40	65.89
—270	6.56	67.02
	100.0	62.55
Froth product, 53.3 pct by weight		
—100 + 150	37.48	2.74
—150 + 200	43.34	4.35
—200 + 270	15.78	15.00
—270	3.49	55.97
	100.00	7.18

Table III. Screen Analyses of Flotation Products with Pearl Starch, 0.04 Lb per Ton

Screen Size, Mesh	Wt, Pct	Iron, Pct
Concentrate, 47.6 pct by weight		
—100 + 150	35.57	59.21
—150 + 200	36.13	63.07
—200 + 270	23.91	65.49
—270	4.39	65.81
	100.00	62.41
Froth product, 52.4 pct by weight		
—100 + 150	39.68	3.39
—150 + 200	43.46	3.95
—200 + 270	14.92	9.11
—270	1.94	51.78
	100.00	5.43

of the coarse fractions of the froth products showed that most of the iron occurred as locked grains.

During the tests, intense flocculation of the quartz occurred after the addition of the collector. It is therefore possible that most of the iron mineral particles carried into the froth were those trapped during flocculation. The possibly beneficial results of adding a dispersant were not studied at this time.

To obtain an effective separation of the iron oxides from the quartz by cationic flotation, it is necessary to maintain the iron oxide surface in a hydrophilic condition and that of the quartz hydrophobic. If laurylamine acetate alone is added, the quartz will be floated, but unfortunately, because of their capability of adsorbing the collector, a considerable quantity of iron oxides will appear in the froth. The function of the starch or of related substances as an additive is to prevent, or at least to minimize, the adsorption of the collector on the iron mineral surfaces so that a higher degree of selectivity may be realized.

Cooke and co-workers⁷ have shown that starches selectively coat hematite in preference to quartz. The mechanism by which the starch inhibits collector coating is not clearly known. However, because of the presence of large numbers of OH groups in the starch molecule, mineral surfaces coated with starch should be hydrophilic.

To evaluate the selectivity and the effectiveness of the starches and starch products tested as iron

oxide depressants, it is necessary to set a specification on both the grade of the iron concentrate produced and on the iron recovery. The quantities of various starches and starch products to produce iron concentrates containing 8.5 and 12.0 pct silica respectively with iron recovery greater than 90 pct were determined with the aid of Figs. 3 and 4 and the silica assay of the iron concentrates.

Table IV. Quantity of Starch to Produce Specified Grade Fe Concentrates at Fe Recoveries Greater than 90 Pct

Type	Starch	Grade of Fe Conc.		
	Lbs per Ton	SiO ₂ , Pct	Fe, Pct	Fe Recovery, Pct
Pearl	0.055	8.5	61.5	92.5
A	0.075	8.5	60.5	94.5
Gum 3502	0.105	8.5	60.5	93.5
Dextrin 156	0.650	8.5	60.0	92.5
B	0.050	12.0	57.5	94.0
Pearl	0.075	12.0	59.0	95.0
A	0.090	12.0	58.0	95.5
Gum 3502	0.150	12.0	58.0	96.0
Dextrin 156	1.020	12.0	58.0	93.0

Table IV shows that Pearl starch, A-fraction starch, Gum 3502, and Dextrin 156 can produce iron concentrate containing 8.5 pct quartz with an iron recovery ranging from 92.5 to 94.5 pct. These substances can therefore be classified as selective iron oxide depressants. B-fraction starch is not included since it produces inferior grade iron concentrate. If the effectiveness of the starches and starch products as selective depressants is based on the quantity of their addition to bring about the specified concentration, then Pearl starch is the most effective depressant tested, followed respectively by A-fraction starch, Gum 3502, and Dextrin 156.

Summary and Conclusions

1—The following starches and starch products, in order of decreasing metallurgical effectiveness, are selective depressants for iron oxide minerals: Pearl starch, A-fraction starch, Gum 3502, and Dextrin 156.

2—B-fraction starch gives poor selectivity as an iron oxide depressant and dextrose has no effect, beneficial or otherwise.

3—The optimum pH for flotation at laurylamine acetate addition of 0.26 lb per ton ranges from 6 to 8.

Acknowledgments

The authors express thanks for the assistance of Henry H. Wade of the Mines Experiment Station, University of Minnesota, Vernon E. Bye and associates of the same institution, the Corn Products Refining Co., and Armour and Co.

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